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Philadelphia Meeting of the American Electro- chemical Society

The admirable program which has been arranged for the Philadelphia meeting should give a new strong impetus to the American Electrochemical Society. The professional and social features of the meeting are exceedingly well balanced, while the program of papers to be read and discussed covers such a wide variety of subjects that there should be enough of interest for everybody, be they engineer, scientist or business man. We trust that a very large attendance will be the reply from the membership at large to the call sent out from the old city of Benjamin Franklin.

Metallurgical Calculations

The article on the electrometallurgy of iron and steel, published in our present issue, concludes the second part of Prof. J. W. Richards' Metallurgical Calculations. The first part, which has already appeared in book form, dealt with the general principles of chemical and metallurgical calculations. The second part, which is now concluded, and which will also be soon available in book form, was entirely devoted to the metallurgy of iron and steel. The third part will relate to the non-ferrous metals. The first instalment, to be published in our next issue, will deal with the metallurgy of copper.

To Every Action there is an Equal and Opposite Action

The recent bad break in the prices of railroad and industrial securities on the New York stock market was not altogether unexpected by the average shrewd outside observer. It was largely a panic among rich speculative pools, which have long been discounting possible prosperity and have had no idea that a storm could ever come to a sea for so long time calm. The jar that disturbed the super-saturated solution of prosperity was undoubtedly the attitude of President Roosevelt on the railroad question. But this was an incidental cause, and the real causes were much deeper-seated. Most direct was the attitude of large railway magnates to the public, expressed in such actions as the sudden declaration of an increased dividend for speculative purposes and also in a flippant and breezy unconcern for the rights of the people disclosed in an official investigation. A more far-reaching, if direct, cause is the vast sums tied up by the public in speculative enterprises, as mining companies, real estate ventures and small industries and street railways. These commitments, though often small individually, will total up enormously in the aggregate. They have diverted large streams of investment funds from the financial centers to the smaller bankers and promoters. And it is not without the substance of fact that the investing public are turning for financial advice in distrust from the large men to the smaller men, whom they know more closely.

One of the further causes is a widespread extravagance of all classes of our people. Had this country the thrifty habits of

France, the ready cash resources of the New Yorker would soon eclipse those of London, Paris and Berlin combined. But it seems impossible with the great influx of money for our people to follow the ways of the habitual saving. Another strong precipitant of market prices of stocks was the increase in the rate of interest. As we have pointed out, the advance in industry, due to modern applied science, has increased greatly the productivity of capital. Hence capital exacts high returns for interest. Interest rates had risen last Fall to such a point that it seemed impossible for stocks to stand at quotations returning only two-thirds of the normal borrowing rate. This year, also, like 1903, was a pre-presidential year, and large retrenchment was in order until our political house-cleaning was over. In several other respects 1907 resembles 1903, and it would be surprising did we not have a recession in business in the Summer and Fall somewhat analogous to the recession in 1903. We are inclined to think that this will be much less serious, for the diffusion of accurate financial intelligence is much better now than then. Prices of commodities are now much too high, especially of copper and lead, and these must be reduced to a normal basis. The action of the United States Steel Corporation in keeping the prices of its products 8 per cent below those of 1902 was extremely wise, for the reaction will be less. Taken in its entirety, the financial and business situation exhibits no distressing symptoms. This country is growing richer very fast—possibly as fast in times of recession as in a boom. As we remarked before, every sane man is a bull on America's industrial future if the future be deferred long enough.

The Cost and the Price of Copper

In valuing the ore reserves of a mine, a certain set price must be put on the recovered metal after deductions are made for metallurgical losses, cost of mining, freight and treatment. As the development of a mine should be a commercial proposition, it is necessary to make some accurate forecast of the average price of a metal for some future period. This should be based on the average cost of production of the largest producers with proper allowances. In order to induce capital to embark in a mining enterprise there should be a large profit on the "turn-over." What this figure should be is impossible to state, for there is no psychological balance to weigh human avarice. But reasoning on the facts, it would appear that 50 per cent is a fair approximation. That is, a mining and smelting business, doing on the average a business of \$12,000 a day, should net \$4,000 a day. The profit on the investment, of course, is much less, for there is a monopolistic value given to the mine which forces this down to about 10 or 12 per cent.

To get the base figure for the cost of copper we can be sure that in 1902 and 1903 very few of the big copper companies were making much money at 11-cent copper. Most of them were paying dividends, but these were often really partly paid out of the capital, for sufficient was not spent in new work. By an accurate system of accounting this was really paid out of capital. In the past five years advances in the art of metallurgy have possibly slightly decreased the cost of making copper. The advantage so gained would have been larger had not the price of labor and supplies risen greatly. Prob-

ably on the average the cost of producing copper and transporting it to New York is now in the Butte district slightly over 9 cents per pound. In the Michigan copper mines the figure is a little higher, probably 9½ cents to 10 cents per pound. In Arizona the cost is much less, and hardly averages more than 7½ cents. One large copper mine of Mexico that formerly made copper for 7 or 8 cents, had a cost of 14 cents. This, however, was due to gross mismanagement.

* * *

When freight, commissions, refining charges and development work are taken into consideration we believe that 10 cents is a fair average figure for bed-rock cost of "laying down" in New York refined ingot copper. Allowing 50 per cent profit on the cost of production, but none too high for the mining business, we have an average New York price for the future of 15 cents per pound. This is a round number which is conservative for estimates on the value of ore reserves. It is, however, not improbable that future development of the electrical industry, especially the electrification of the great railroad systems of America, will give at times an artificially high value, because the demand occasionally may increase faster than the means of supplying this demand. It would appear that the copper market has just passed through one of these periods of abnormally high prices and is liable now to return to a figure more equitable alike to producer and consumer.

The Open-Hearth Steel Process

The open-hearth steel statistics just presented by the American Iron and Steel Association show a production in 1906 of 10,970,998 gross tons of ingots and castings. The record of production is shown below, beginning with 1896, the first year for which separate statistics for the basic and acid processes were ascertained:

UNITED STATES PRODUCTION OF OPEN-HEARTH INGOTS AND CASTINGS, GROSS TONS.

	Basic.	Acid.	Total.
1896.....	776,256	522,444	1,298,700
1897.....	1,056,043	552,628	1,608,671
1898.....	1,569,412	660,880	2,230,292
1899.....	2,080,426	866,890	2,947,316
1900.....	2,545,091	853,044	3,398,135
1901.....	3,618,993	1,037,316	4,656,309
1902.....	4,496,533	1,191,196	5,687,729
1903.....	4,734,913	1,094,998	5,829,911
1904.....	5,106,367	801,799	5,908,166
1905.....	7,815,728	1,155,648	8,971,376
1906.....	9,649,385	1,321,613	10,970,998

It would not be easy to find a parallel to the record shown above. In not one year has the production of open-hearth steel shown a decrease, while the production of basic steel has advanced very rapidly. Production by the acid process has less steadily increased, having taken backward steps in 1900, 1903 and 1904, the 1902 record not being broken until 1906. In 1896 nearly as much steel was made by the acid as by the basic open-hearth process; in 1900 there was only one-third as much; in 1906 there was less than one-seventh as much.

* * *

Three causes have chiefly contributed to the growth of the basic relative to the acid open-hearth process. In the first

place, better steel is being made, in the circumstances, now than formerly. If we may be pardoned for the expression the basic open-hearth furnace was at first hailed chiefly as a good thing in which to utilize junk, now it is a means of making steel. The amount of junk available has not increased nearly as rapidly as has the output of steel, better pig iron, on the average, is being used, and much of the scrap used is of the finest quality, being new material. This scrap is, in a sense, made for the purpose, a fact we shall recur to later. In the second place, engineers have quite largely learned to call for steel rather than processes. It is quite certain that a few years ago many engineers were specifying against basic steel merely because they were doubtful of their own ability to specify, purely on chemical and physical grounds, for the quality they desired to be furnished them. Now they are able to define what they want in a more scientific way. In the third place, Bessemer pig iron, which must be used in the acid open-hearth process, is growing scarce, while there is plenty of basic pig, low in phosphorus relative to the basic pig at first employed, but still outside the Bessemer limit of 0.10.

* * *

The knowledge of how largely old material was employed in the basic open-hearth furnace in its early years, through the late nineties, the charges running up to 80 per cent scrap in many cases, and that the outcome of old material has quite fallen short of equaling the rapid increase in basic open-hearth steel ingot production, while various processes have been adopted which are claimed to eliminate the need for scrap, does not lead to an understanding of the statistics of basic open-hearth steel ingot production proportionate to the production of basic pig iron. In 1896 the production of basic pig iron constituted 43.3 per cent of the production of basic open-hearth steel ingots; in 1897 the percentage was 52.7 per cent, the highest shown; it fell to 40.0 per cent in 1901, and has since then increased again, being 48.6 per cent in 1904, 52.5 per cent in 1905 and 52.1 per cent in 1906. At one time Bessemer pig iron was used to a small extent in this process, but it is not so now, and the limited quantities of charcoal pig iron used are negligible. With the relatively reduced quantity of old material available, and the use at many plants of the so-called non-scrap processes, why is almost double as much steel produced as pig iron?

* * *

The statistics of Bessemer and open-hearth steel ingot and casting production are correct, in their way, but they are strictly what they are stated to be, and nothing else. Neglecting the comparatively small tonnage of castings included, they are the statistics of the production of steel ingots. They are not really statistics of the production of steel at all. To the lay mind a steel ingot is made merely that it may be rolled into a finished product for the market, with a loss in scale and scrap which the manager, dealing with a valuable material, will endeavor to keep down to the lowest proportion. To the manager, however, the steel ingot is produced as something from which he can select a portion to roll into finished product and sell, leaving a large portion to go into the open-hearth furnace. The bulk of the Bessemer steel scrap does not return to the converter, but goes to the open-hearth furnace, while in a sense the open-hearth material is making round trips from the furnace to the shear and back again. Between the

statistics of ingot production and the statistics of rolled steel production there is a gap which is bridged by scrap, not made as a necessary incident to the rolling, but as a necessity to the production of good steel, and a convenience to the open-hearth furnace.

* * *

The 1905 statistics shows a production of ingots (excluding castings) of 10,919,272 gross tons by the Bessemer process, 8,444,836 tons by the open-hearth process, and 99,072 tons by the crucible and other processes, making a total of 19,463,180 gross tons of steel ingots, while the total production of all rolled steel was reported at 14,780,025 gross tons, leaving a spread of 4,683,155 tons. In that year the difference between basic pig iron and basic ingot and casting production was 3,700,000 tons; a large part of this was made up of scrap figuring in the above tonnage. Another and smaller part was, of course, made through the use of ore. It is well, therefore, to remember that the statistics of ingot production are literally that, and not statistics of steel production. In no current literature do we ever see this distinction made, yet it is a most important one, seeing that the difference now runs into millions of tons. Strictly speaking, steel is produced when the necessary elimination of impurities, whether in pig iron, ore or scrap, has been made. It is not altogether making steel to pour an open-hearth heat, cast it in ingots, weigh the ingots, and then send a considerable part back to the furnace, nor is it altogether making steel to cast a Bessemer ingot, cut part of it off, melt it over again in an open-hearth furnace and then weigh the second ingot of which it makes part.

* * *

So, in a measure, the open-hearth steel statistics with which we are dealing are misleading. The ingot and casting statistics for 1906 show 12,275,253 gross tons attributed to the Bessemer and 10,970,998 tons attributed to the open-hearth process; the actual work of the two processes was not in this proportion, nor was the total tonnage of finished steel actually produced and sold at all commensurate with these figures. These are really merely the records of the ingot scales. Incidentally, we are inclined to think those scales are hardly as accurate as those whose records go to the invoice clerks. While sight must not be lost of the fact that these open-hearth steel ingot figures are very generally misunderstood, it is true that the basic open-hearth process is growing rapidly. The open-hearth ingot and casting figures for 1906 show but 1,304,255 gross tons less than the Bessemer figures. Last month we noted that the Bessemer production has stopped its rapid growth, and will in future grow but slowly if at all. The open hearth process, on the other hand, continues to grow rapidly. Without attempting to catalogue all the new erection of open-hearth furnaces, it may be remarked that the United States Steel Corporation is now building 18 furnaces at Duquesne, 12 at Youngstown and 56 at Gary, Ind., a total of 86 furnaces at three points. These furnaces are rated at 50 to 60 tons capacity, and will make, using direct metal, 16 to 18 heats per week, or 75,000 to 80,000 tons of ingots per week, or not far from 4,000,000 tons per year. As they will come in at intervals during the latter part of this year and early next year, they will easily swing the preponderance, in point of ingot tonnage, from Bessemer to open-hearth, giving the open-hearth process a greater lead over Bessemer in 1908 than the latter had over open-hearth in 1906.

Program of Philadelphia Meeting of American Electrochemical Society.

The official programme of the eleventh general meeting of the American Electrochemical Society, to be held in Philadelphia on May 2, 3 and 4, is so very attractive that the evidently great efforts of the local committee should be rewarded by an unusually large attendance.

The meetings will be held at the University of Pennsylvania, in the Lecture Theater of the John Harrison Laboratory of Chemistry, at the northeast corner of Thirty-fourth and Spruce Streets. The headquarters for registration will be in the same building, where members and guests introduced by members will receive badges.

The first meeting will be opened on *Thursday* morning, May 2, at 10 A. M., by an address of welcome by Dr. Edgar F. Smith, vice-provost of the University of Pennsylvania. At the following business meeting the reports of officers and directors will be presented and the retiring president, Mr. Carl Hering, will then read his presidential address. Then follows a professional session, devoted to the reading and discussion of papers. The meeting will adjourn at 1 P. M. Luncheon will be served at Houston Hall on the University grounds, the University being hostess.

In the afternoon, at 2 P. M., an inspection will be made of the chemical and the engineering laboratories of the University of Pennsylvania. At 3 P. M. the United Water Improvement Co., Thirtieth Street, below Locust, will be visited, where the commercial production of ozone for water purification will be shown. This is to be followed by a visit to the plant of the John F. Lewis & Bros. Co., where the manufacture of white and red leads will be shown. The factory is at 2570 East Thompson Street. For the evening of Thursday a theater party has been arranged at the new Keith's Theater in Chestnut Street.

On *Friday*, at 9:30 A. M., the second session, devoted to the reading and discussion of papers, will be held. At 1 P. M. the members and guests will meet at Chestnut Street wharf on the city fireboat "Ashbridge," for a private trip on the Delaware River, tendered to the members of this Society by the city of Philadelphia, through the kindness of the Honorable Mayor and the Director of Public Safety. The boat will leave at 1 P. M. sharp, arriving at Tacony at about 1:50, where the party will visit the Diston Saw Works, at which the Colby electric induction furnace for smelting steel is expected to be shown in operation. Thence to the Lardner's Point Pumping Station, where very large pumping engines will be shown. Thence to the Frankford station of the Philadelphia Electric Co., where Mr. C. J. Russell will show working models of the following electric furnaces: The Colby induction furnace, the Héroult type of furnace, an annealing and tempering furnace with fused barium chloride and a melting furnace using kryptol.

Return to the boat at 3 P. M. sharp, thence down the Delaware, passing the Philadelphia and Camden water fronts with all their large industries, including Cramp's ship yard, the New York Shipbuilding plant, the By-Product Coke Ovens, sugar refineries, League Island Navy Yard, etc., and arriving at Gloucester at 4 to 4:30 P. M., to visit the Welsbach Light Co's works, where the manufacture of mantels and other interesting processes will be shown.

After having met again on the boat, at 5:45 P. M., the party will arrive at Washington Park at 6, where the landing of the shad nets may be seen, which will be followed, in fine Philadelphia style, at 7:30 P. M., by a planked shad dinner. This informal dinner will take the place of the usual banquet.

On *Saturday* morning, May 4, 9:30 A. M., the third session, devoted to the reading and discussion of papers, will be held. The meeting will adjourn at 12:30. Then follows a demonstration of exhibits. There will be an exhibition of apparatus

of special interest to electrochemists in the Lecture Theater of the Chemical Laboratory. These exhibits will be open for inspection during the whole meeting, but the demonstrations will take place after the Saturday session. At 1:30 P. M., the members and guests will take lunch at Houston Hall, as guests at the University.

In the afternoon a visit will be paid to the United States Mint, at Seventeenth and Spring Garden Streets. Here the electrolytic processes for the refining of gold and silver will be shown under the direction of Dr. D. K. Tuttle and Mr. Leonard Morgan. The party will divide into groups of not more than ten to fifteen.

The following papers will be presented at the three sessions. *Thursday* morning session, immediately after the presidential address of Mr. Carl Hering:

1. Some Efficiency Considerations of an Electrolytic Cell, W. R. Mott.
2. Changes of Concentration and Migration Velocities, C. J. Reed.
3. The Work Done in Electrolysis, J. W. Richards.
4. The Present Status of the Thermodynamic Scale, E. Buckingham.
5. Determination of Fixed Points in Pyrometry, G. K. Burgess.
6. Bomb Calorimeter for Measuring the Heat of Combustion of Substances Giving Solid Oxidation Products, Henry Noel Potter.
7. On the Density, Electrical Conductivity and Viscosity of Certain Fused Salts and Their Mixtures, H. M. Goodwin and R. D. Mailey.
8. Electrolytic Pickling of Steel, C. J. Reed.
9. Electric Reduction of Titaniferous Iron Ore, Gustave Gin.
10. Recent Developments in Electrolytic Analysis, E. F. Smith.

Friday morning session:

11. Illustrated lecture by invitation of the Board of Directors, by Dr. C. P. Steinmetz, on the Conductions of Vapors and Gases.
12. The Electro-Metallurgy of Zinc and Its Relation to the Present Practice, W. McA. Johnson.
13. Electrodeposition of Zinc, R. C. Snowdon.
14. A Practical Limitation of Resistance Furnaces; the "Pinch" Phenomenon, Carl Hering.
15. The Action of Carbon on Magnesia at High Temperatures, O. P. Watts.
16. Reply to paper by E. A. Sperry, "Electrochemical Processes as Station Load Equalizers," Trans. IX., 147, John Meyer.
17. Relative Cost of Power, Charles E. Lucke.
18. Surface Properties of Aluminium and Zinc, W. J. Hammer.
19. The Helion Lamp, H. J. Parker.

Saturday morning session:

20. Electrolytic Corrosion of Brass (1906 award), A. T. Lincoln.
21. Rapid Determination of Electrolytic Resistance, C. F. Burgess.
22. C. F. Burgess (title to be announced).
23. Polarization Voltages of Silver Nitrate Solutions, J. A. Wilkinson and H. A. Gillett.
24. The Electrolytic Deposition of Nickel-Zinc Alloys, E. P. Schoch and Alcan Hirsch.
25. A closed Electric Furnace for Reducing and Distilling Metals from Their Ores, E. R. Taylor.
26. Evolution of Modern Methods of Parting and Refining, D. K. Tuttle.
27. Granular Carbon Resistance Furnace, S. A. Tucker.
28. Electric Tube Furnace for Temperature Measurements, S. A. Tucker.
29. Influence of Strain on the Corrosion of Iron, W. H. Walker.

30. On Carbon for Electrometallurgy, F. A. J. FitzGerald and J. Forssell.

31. The Titanium Arc, R. Fleming.

The executive local committee consists of Dr. E. F. Smith, chairman; Mr. Carl Hering, vice-chairman; Mr. S. S. Sadtler, 39 South Tenth Street, Philadelphia, secretary; Mr. George Breed and Dr. J. W. Richards.

The chairman on the committee on excursions is Dr. J. W. Richards; the chairman of the entertainment committee, Mr. George Breed; the chairman of the University committee, Dr. E. F. Smith.

The ladies' committee consists of Mrs. George Breed, Mrs. C. J. Reed, Mrs. J. W. Richards, Mrs. S. S. Sadtler, Mrs. P. G. Salom and Mrs. E. F. Smith.

The Iron and Steel Market.

Developments in the trade in April have been quite in line with the views expressed in last report, that the near future of the market is entirely assured of satisfactory conditions, while the two or three closing months of this year, and the entire year 1908, are more or less in doubt. It does not follow that there is assurance of even the mildest kind of a depression in 1908; usually the iron trade cannot see so far ahead; occasionally it can, but it happens that this is not one of the occasions.

There has been a continuance of heavy specifications in all finished steel products, according to all accounts, and the guarantee of full activity for all the mills through the year is questioned only by those who profess to see a much greater willingness than usual on the part of producers to take the public into their confidence as to the state of their order books.

Pig iron for second half delivery has strengthened during the month, some large purchases of steel-making pig being made for second-half delivery, while, a matter of still more importance, some consumers who were already covered through the third quarter have come into the market and bought fourth-quarter iron at full prices.

The statistics of open-hearth steel production in 1906 were announced about the middle of the month, and showed a total production of ingots and castings of 10,970,998 gross tons, of which 9,649,385 tons were made by the basic and 1,321,613 tons by the acid process, a gain in open-hearth steel over 1905 of practically 2,000,000 tons, the second largest gain, in point of tonnage, ever shown in one year by the open-hearth process.

No important change in prices of finished steel products has occurred during the month; iron bars are slightly lower, pig iron is stronger and billets for Pittsburg delivery are higher.

PIG IRON.

Furnaces early in the month receded from their position of demanding \$21.50, valley, for second-half Bessemer, making light sales at \$21.00, valley. This move did not encourage a waiting policy, as such moves usually do, but was followed by heavier sales at this price, and the market has become thoroughly established on Bessemer and basic at \$21.00, valley, for any delivery during the second half, with prospects that regular consumers will easily absorb all the available tonnage, leaving none for such occasional buyers as the United States Steel Corporation, which buy only to supplement their own production or to support the market. Among Bessemer sales during the month were the following: Two contracts aggregating about 70,000 tons on the sliding scale basis, over half of this being in one contract for second-half delivery, the balance on a contract for twelve months beginning with April; 157,000 tons to the Youngstown Sheet & Tube Co., being 10,000 tons for June at \$22.00, valley, and 147,000 tons for second half at \$21.00, valley, 122,000 tons being furnished by the Bessemer Furnace Association, 15,000 tons by the United

Iron & Steel Co. (which recently absorbed the Cherry Valley properties), and 10,000 tons by the Andrews & Hitchcock Iron Co., 2,500 tons for third quarter and bought later by the same interest, 2,500 tons for fourth quarter, at \$21.00, valley, and various lots aggregating perhaps 10,000 tons for deliveries in the second quarter at \$22.50 to \$23.00, valley. Sales of basic included 6,500 tons for third and 5,000 tons for fourth quarter, bought at different times by one consumer, at \$21.00, valley, and a sliding scale contract covering a small monthly tonnage for a period of months. In foundry iron, sales have been lighter. Some Buffalo furnaces are reported to have sold for the first quarter of 1908 at good prices; Eastern and valley furnaces have sold moderate tonnages for second half, the Eastern furnaces at slightly lower prices than they had been asking, and the valley furnaces at slightly higher prices than could have been done a month earlier. The middlemen who were offering Southern iron for second half at \$18.00, Birmingham, presumably with the connivance of the furnaces, who were themselves asking \$18.50, have withdrawn, and the Southern market is firm at \$18.50 for second half. The 25-cent advance in freight rates to the Ohio River finally went into effect April 1, so that delivered prices, on an \$18.50 basis, are \$21.75, Cincinnati; \$23.35, Pittsburg, and \$22.85, Chicago. In the valleys No. 2 foundry for second half is pretty firm at \$22.00, valley, or \$22.85, Pittsburg or Cleveland. For any delivery prior to July furnaces are obtaining premiums of from \$1.00 to \$3.00 a ton.

STEEL.

No billets are obtainable from Pittsburg, producers there requiring all their product for their own finishing mills, and the Pittsburg market is the price f. o. b. other points which are offering, plus the freight. For instance, a sale of 1,000 tons of Bessemer billets was made at about \$29.50 f. o. b. Youngstown, or about \$30.50, delivered Pittsburg, and a sale of open-hearth billets at about \$31.70, Pittsburg, the steel coming from a point taking a freight rate between \$1.50 and \$2.00. Sheet bars are nominally \$30.00 f. o. b., Pittsburg or Youngstown, but for prompt delivery mills are endeavoring to obtain \$31.00, while it is reported a prominent consumer bid \$30.00, mill, for 15,000 tons for fourth quarter without obtaining the material.

RAILS.

It is reported that the Tennessee Coal, Iron & Railroad Co. has sold 46,000 tons of open-hearth rails for 1908 delivery at \$30.00, Birmingham. For 1906 delivery this company sold its rails at \$28.00, Birmingham, the same price, f. o. b. mill, as is obtained for Bessemer rails; for 1907 it made an advance of \$1.00 a ton, and if the present report is correct another dollar advance has been made. Open-hearth rails might, perhaps, properly command an advance of \$2.00 a ton over Bessemer rails, but as the Tennessee company's sales are made to contiguous roads, which otherwise would have to buy Bessemer at \$28.00, f. o. b. Pittsburg or Chicago, the roads are really not paying so large a premium for the open-hearth product. Nothing is known as to what the United States Steel Corporation will do in the matter of open-hearth rail prices. About January next it will be able to make open-hearth rails at Youngstown, and soon thereafter should begin operations at Gary. The reticence in this matter gives rise to an idea that the price will be governed by commercial considerations rather than considerations of the actual relative value of Bessemer and open-hearth rails. Orders are now being booked for 1908 delivery of Bessemer rails.

FINISHED STEEL.

Specification have been heavy in all lines, but there has not been a great deal of new buying. Some contracts for shapes, bars, plates, etc., expired April 1 and have not been renewed, but there were heavy unfilled specifications against them. The agricultural implement makers, not being given any concession for their annual contracts on steel bars from July 1, as was done a year ago, have refrained from contracting, but still

have large tonnages due them. Plates can be had from some of the smaller mills for early delivery at less than former premiums, but the large mills, which sell only at the regular price, are filled up farther ahead than formerly. Structural business is improved, mills having specifications on books for solid runs of from two to three months. Regular mill prices, on which premiums are frequently charged for early delivery, remain as follows:

Structural shapes, \$1.70 for beams and channels, 15 inches and under, angles and tees; \$1.75 for tees; \$1.80 for beams and channels over 15 inches.

Plates, \$1.70 for tank quality, $\frac{1}{4}$ inch and heavier, 100 inches wide and less; flange, \$1.80.

Merchant steel bars, \$1.80, base.

Common iron bars, \$1.80, delivered Pittsburg; \$1.65, f. o. b. Pittsburg, for Western delivery; these prices have been shaded occasionally in the past month.

Sheets, 28 gauge, \$2.60 for black, \$3.75 for galvanized.

Tin plates, \$3.90 for 100-pound cokes.

Dedication of the United Engineering Societies Building.

The dedication exercises of the monumental and beautiful United Engineering Building took place on April 16 and 17, according to the program which was given in full in our last issue. In connection with these dedication exercises, meetings were held by the three founder societies, the American Institute of Electrical Engineers, the American Institute of Mining Engineers and the American Society of Mechanical Engineers.

The Engineering Societies Building is located on West Thirty-ninth Street, New York, north side, between Fifth and Sixth Avenues. The frontage is 125 feet, the depth 90 feet. The land has been bought with money brought up by the three founder societies, the cost being \$502,000, while for the erection of the building, Mr. Andrew Carnegie had given the sum of \$1,500,000. The holding corporation is the United Engineering Society, which acts as trustee for the three founder societies.

The building is $13\frac{1}{2}$ stories high. The exterior is built of limestone up to the auditorium floor, and of gray mottled brick and terra cotta above. On the first and ninth floors passages connect the building with the new club house of the Engineers' Club.

The first floor contains the main foyer, writing room, smoking room, etc. Immediately above it is the coat room. The main auditorium, which is exceedingly handsome and fully meets all requirements, extends through the third and fourth floors. On the fifth and sixth floors there are seven smaller lecture rooms.

The seventh and eighth floors have been reserved for the "associated societies," among which are the National Electric Light Association, the New York Electrical Society, the Society of Naval Engineers and Marine Architects and others. The next three floors are occupied by the three founder societies—the ninth floor by the American Institute of Mining Engineers, the tenth by the American Institute of Electrical Engineers, and the eleventh by the American Society of Mechanical Engineers. Finally, the twelfth and thirteenth floors contain the libraries of these three founder societies. They form the nucleus of what is expected to become the greatest engineering library of the world.

On the first day of the dedication exercises, Tuesday, April 16, Mr. Charles W. Hunt was the presiding officer. Congratulatory letters from President Roosevelt and President Diaz of Mexico were read. Mr. Charles F. Scott gave an outline of the history of the building. Mr. E. E. Olcott spoke for the United Engineering Society, after having received the key of the building.

Then spoke Mr. Carnegie. He emphasized the need of co-

operation as the keynote of universal brotherhood. He expressed his conviction that the world is getting better every day. To-day is better than yesterday. To-morrow will be better than to-day. So there is no need to worry for the future.

The closing address of the day was made by President Arthur T. Hadley, of Yale, on the professional ideals of the twentieth century. "A hundred years ago we might have had a building in honor of theologians, of lawyers or of physicians." In the twentieth century the engineers got their home. "Yours is the proud boast of having in one brief century established a science as the arbiter of the material affairs of mankind, and of having enforced her worship upon a world once reluctant but now gloriously admiring." What then is still to be done? "It is not enough to have technical training. It is not enough to know the special sciences on which the practice of a profession is based. A man ought to have clear conceptions of the public service which his profession can render, and the public duty which its members owe. Thus, and thus only, can the engineer, the lawyer, the physician, or a member of any of the learned professions rise to the full dignity of his calling." Besides the technical standard there is the ethical standard and professional men must learn to live up to the latter as well as to the former. On the evening of Tuesday a reception was held which was exceedingly well attended.

At the exercises of Wednesday, Mr. John W. Lieb, Jr., presided, and the three presidents of the founder societies, Dr. Samuel Sheldon, Dr. F. R. Hutton and Dr. John Hays Hammond, delivered addresses.

Then followed a great many representatives of institutes of learning and engineering societies in this country and abroad. Mr. T. C. Martin, as president of the Engineers' Club, made a hit by presenting the greetings from the club in a clever little poem: "To our fellow engineers, who've been looking many years, up-town and down, for a home whose rental wasn't high; we members of the club, which has also felt the rub, bring greetings from our little house nearby." He invited all to come over "when weary of the jargon of *api* and of *argon*," for "in the club we are human, not forgetting there is woman." The "little house" is the magnificent new building of the Engineers' Club, which has also been donated by Mr. Carnegie.

CORRESPONDENCE

Melting Points of Elements.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—The table of melting points of elements in your February issue, page 48, contains some errors.¹

According to the latest investigation of Holborn and Valentiner (*Annalen der Physik*, Vol. 22, 1907, pp. 1 to 48) the melting point of platinum is $1,790^{\circ}$ C. The value $1,710^{\circ}$ which has heretofore been considered to be the correct one, was based on an erroneous extrapolation for the thermo-couple. Thus we have now come back approximately to the old value found many years ago by Violle.

The figure 800° for magnesium is all wrong; it melts at 633° (Heycock and Neville, 1895).

The latest investigation of Holborn and Valentiner has also given an exact value for the melting point of palladium. It is $1,575^{\circ}$.

Of less importance are the following points: Calcium, 800° (not 760°); nickel, $1,484^{\circ}$ (not $1,427^{\circ}$); tin, 241° (not 224°).

It is a pleasure for me to state here that your journal has won a great many friends in Germany.

KURT ARNDT.

Technische Hochschule, Berlin, Germany.

¹ See also page 76 of our March issue.—Editor.

Metallurgical Calculations.

By J. W. RICHARDS, PH. D.

Professor of Metallurgy in Lehigh University.

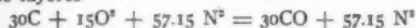
The Electrometallurgy of Iron and Steel.

Electrical methods may enter into the extraction of a metal from its ores either as electrolytic or as electrothermal processes. Electrolytic processes are those in which the electric current is used for its electrolytic action, *i. e.*, for its electrical decomposing and depositing properties; electrothermal processes are such as use the current merely as a source of heat, to furnish the sensible heat and high temperature necessary for melting materials or for bringing about chemical reactions. So far electrolytic processes have entered the metallurgy of iron only as used by Burgess for electrolytically refining nearly pure iron in an aqueous electrolyte and depositing chemically pure iron; the electrolysis of fused iron salts has not been practically utilized. Up to the present, electrothermal processes are in commercial use for melting together wrought iron and cast iron to make steel, also for keeping cast iron melted while its impurities are being extracted by oxidation; the electrothermal reduction of iron ores to cast iron has been proved technically possible, and may in some places prove commercially practicable.

ELECTROTHERMAL REDUCTION OF IRON ORES.

If the electric current is used to furnish the heat energy necessary to reduce iron ore, it cannot displace the reducing agent-carbon. In ordinary blast furnace practice the carbon is first burned to provide the heat necessary to smelt down the pig iron and slag, and the product of this incomplete combustion—CO—abstracts oxygen from the ore. The two equations are practically:

At the tuyeres—



Reduction:



These equations show us that to produce 8Fe = 448 parts, 30C = 360 parts of carbon is the minimum necessary, which is first burned to CO at the tuyeres, and then the producer gas thus formed (N² and CO) reduces the iron oxides above. If the heat for fusion is furnished electrically, the first combustion is unnecessary, all blowing in of air is dispensed with and the reaction taking place is:



And we have 28 Fe = 1,568 parts reduced by 30C = 360 parts, a consumption of less than one-third as much carbon as is required in the blast furnace. The operation consists, therefore, in mixing iron ore and carbon so that for every part of iron present about 0.25 parts of carbon is present, using the proper quantity of limestone or other material to flux the gangue of the ore to a fusible slag, and then furnishing electrically the heat necessary to cause the chemical reaction, melt down the resulting iron and slag, and supply radiation losses. The gases resulting from this electrical reduction are combustible, just as the gases from the blast furnace, and since there is no blast to be heated they can very well be utilized to preheat the charges coming into the furnace, and thus save some of the electrical energy needed.

Problem 73.

A magnetite ore contains:

	Per Cent.		Per Cent.
Fe ² O ³	60.74	MgO	5.50
FeO	17.18	P ² O ⁵	0.04
SiO ²	6.60	S	0.57
Al ² O ³	1.48	CO ²	2.05
CaO	2.84	H ² O	3.00

It is to be mixed with pure carbon (charcoal fines) and a

suitable flux; the fixed carbon being 0.25 per cent of the iron present; the flux silica sand, so as to make a slag with 33 per cent of SiO². Neglect the ash and assume 10 per cent of moisture in the charcoal. Assume also:

(a) The pig iron to contain 4 per cent C, 3.5 per cent Si, 92.4 per cent Fe.

(b) The slag and pig iron to contain at tapping 600 and 400 Calories of heat respectively.

(c) The heat losses by radiation, etc., to be 30 per cent of the total heat requirement of the furnace.

(d) The hot gases to escape at 300° C.

(e) The iron to be completely reduced into the pig iron.

(f) The sulphur to go entirely into the slag as CaS.

Requirements:

(1) The weights of ore, flux and charcoal dust needed per 1,000 kg. of pig iron produced.

(2) A balance sheet of materials entering and leaving the furnace.

(3) A heat balance sheet of the furnace.

(4) The number of kilowatt days of electrical energy required per metric ton of pig iron produced.

Solution:

(1) The ore must supply 924 kg. of iron. But 100 parts of ore contains iron as follows:

		Kg.
In Fe ² O ³	60.74 × 112/160 =	42.52
In FeO	17.18 × 56/72 =	13.36
	Sum =	55.88

Ore required per 1,000 kg. of pig iron:

$$924 \div 0.5588 = 1,654 \text{ kg.} \quad (1)$$

The slag-forming ingredients from this amount of ore are as follows:

		Kg.
Al ² O ³	1,654 × 0.0148	= 24.6
MgO	1,654 × 0.0550	= 91.0
CaO	1,654 × (0.0284 — 0.0057 × 50/32)	= 1,654 × 0.0184 = 30.4
SiO ²	(1,654 × 0.0660) — (35 × 60/28) = 109.2 — 75 =	34.2
CaS	1,654 × (0.0057 × 72/32)	= 21.2
	Sum =	201.4

If *x* parts of SiO² sand are added to these, the total weight of slag will be 201.4 + *x*, of SiO² in it 34.2 + *x*. And since the SiO² is to be 33 per cent of the weight of slag, then

$$34.2 + x = 0.33 (201.4 + x) \\ x = 48 \text{ kg.} \quad (1)$$

The charcoal dust used must contain fixed carbon equal to 0.25 of the iron present, *i. e.*,

$$924 \times 0.25 = 231 \text{ kg.}$$

And since it is 90 per cent fixed carbon the dust required is:

$$231 \div 0.90 = 257 \text{ kg.} \quad (1)$$

Charges.	Pig Iron.	Slag.	Gases.
Ore.....1654 Kg.			
Fe ² O ³1004.6	Fe..... 703.2	O..... 301.4
FeO..... 284.2	Fe..... 221.0	O..... 63.2
SiO ² 109.2	Si..... 35.0	SiO ² ... 34.2	O..... 40.0
Al ² O ³ 24.6	Al ² O ³ ... 24.6
CaO..... 46.9	CaO..... 30.4	O..... 4.7
MgO..... 91.0	MgO..... 91.0
P ² O ⁵ 0.6	P..... 0.3	Ca..... 11.8	O..... 0.3
S..... 9.4	S..... 9.4
CO ² 33.9	CO ² ... 33.9
H ² O..... 49.6	H ² O... 49.6
Flux..... 48 Kg.			
SiO ² 48.0	SiO ² ... 48.0
Charcoal.....257Kg.			
C..... 231.0	C..... 40.0	C..... 191.0
H ² O..... 26.0	H ² O... 26.0
	1959.0	999.5	249.4
			710.1

(3) Heat available is the heat of oxidation of carbon plus that furnished by the electric current. The charge gives up to the carbon, as shown on the balance sheet, $301.4 + 63.2 + 40.0 + 4.7 + 0.3 = 409.6$ kg. of oxygen. The 191 kg. of carbon burned would take $191 \times 16/12 = 254.7$ kg. of oxygen to burn it to CO, leaving 154.9 kg. of oxygen to burn CO to CO^2 . This would burn $154.9 \times 28/16 = 271.1$ kg. of CO to CO^2 .

The heat of formation of the slag may be taken as approximately 150 Calories per kg. of contained $\text{SiO}^2 + \text{Al}^2\text{O}^3$. The heat of combination of carbon with the iron is a doubtful quantity, which may be taken at 705 Calories per kilogram of carbon. The formation of CaS gives 2,947 Calories per kilogram of sulphur.

Letting the heat furnished by the electric current = x , and neglecting the heat of oxidation of the small amount of electrode carbon consumed, we have:

Heat Available.

		Calories.
Supplied by electric current:	x	
Oxidation of C to CO	$191 \times 2,430 =$	464,130
Oxidation of CO to CO^2	$271.1 \times 2,430 =$	658,770
Formation of silicate slag	$106.8 \times 150 =$	16,020
Formation of CaS	$9.4 \times 2,947 =$	27,700
Formation of Fe^2C	$40. \times 705 =$	28,200
Sum		$= 1,194,820 + x$

Heat Distribution.

	Calories.
Reduction of Fe from Fe^2O^3	
$703.2 \times 1,746$	$= 1,229,790$
Reduction of Fe from FeO	
$221.0 \times 1,173$	$= 259,230$
Reduction of Si from SiO^2	
$35.0 \times 7,000$	$= 245,000$
Reduction of P from P^2O^5	
$0.3 \times 5,892$	$= 1,770$
Reduction of Ca from CaO	
$11.8 \times 3,288$	$= 38,800$
Expulsion of CO^2 from ore	
$33.9 \times 1,026$	$= 34,780$
Evaporation of H^2O from charges	
75.6×606.5	$= 45,850$
Sensible heat in gases, at 300°	
CO $174.6 \text{ kg.} = 138 \text{ m}^3$	
$\times 0.311$	$= 42.9$
CO^2 $459.9 \text{ kg.} = 232 \text{ m}^3$	
$\times 0.436$	$= 101.1$
H^2O $75.6 \text{ kg.} = 93 \text{ m}^3$	
$\times 0.385$	$= 35.8$
	$179.8 \times 300 = 53,940$
Sensible heat in slag 249.4×600	$= 149,640$
Sensible heat in pig iron $1,000 \times 400$	$= 400,000$
Loss by radiation, etc., $0.30 (1,194,820 + x)$	$= 358,450 + 0.3 x$
Sum total	$= 2,817,250 + 0.3 x$

Equating the heat available and accounted for we have:

$$1,194,820 + x = 2,817,250 + 0.3 x.$$

Whence $x = 2,317,760$ Calories.

And the sum total of heat requirement is

$$3,512,580 \text{ Calories,}$$

of which the electric current supplies

$$\frac{2,317,760}{3,512,580} = 0.66 = 66 \text{ per cent.} \quad (3)$$

(4) A kilowatt-day of electricity is equal to

$$0.239 \times 60 \times 60 \times 24 = 20,520 \text{ Calories.}$$

There is, therefore, required per ton of pig iron produced:

$$\frac{2,317,760}{20,520} = 112 \text{ kilowatt days.} \quad (4)$$

This figure might be materially reduced by using the waste gases to warm up the charges entering the furnace. It is also possible that in a properly designed shaft the gases passing out might contain nearly equal volumes of CO and CO^2 , instead of 3CO to 2 CO^2 , as assumed in this problem, from best ordinary blast furnace practice. Any greater utilization of the heat-producing power of the carbon would decrease the electrical energy required; the above calculated value is given as a safe figure for this ore on which to base working calculations.

Problem 74.

At Sault Sainte Marie, Canada, roasted pyrrhotite ore was smelted with the addition of limestone and charcoal fines. The mixture used contained 400 pounds of ore, 110 pounds charcoal dust and 85 pounds limestone. The analyses of each of these materials was:

Roasted Ore.	Limestone.	Charcoal Dust.
Fe^2O^3 65.43	CaO..... 52.00	Fixed C..... 55.90
CuO..... 0.51	MgO..... 2.10	Vol. matter..... 28.08
NiO..... 2.84	Fe^2O^3 0.60	Moisture..... 13.48
SiO^2 10.96	Al^2O^3 0.21	Ash..... 2.54
Al^2O^3 3.31	SiO^2 1.71	
CaO..... 3.92	P^2O^5 0.01	
MgO..... 3.53	SO^2 0.13	
SO^2 3.90	CO^2 43.15	
P^2O^5 0.03		
H^2O 5.57		

Using 165.65 kilowatts effective electric energy for 56 hours 20 minutes, there was produced 7,336 pounds of nickeliferous pig iron and 5,062 pounds of slag, having the following average compositions:

Pig Iron.	Slag.
C 3.05	SiO^2 16.44
Si 5.24	Al^2O^3 13.86
S 0.01	CaO 42.87
P 0.05	MgO 8.80
Cu 0.81	CaS 13.34
Ni 3.94	FeO 0.84
Fe 86.90	Undetermined .. 3.85

Requirements:

(1) A balance sheet of materials entering and leaving the furnace.

(2) A heat balance sheet of the furnace, making necessary assumptions where data is not furnished.

(3) The thermal efficiency of the furnace.

Solution:

(1) The amount of ore used may be calculated either from the iron, the nickel or the copper. The nickel and copper are the easiest to use, because there is supposed to be none of them in the slag, but they are the least reliable, because present in such small amount. The pig-metal contains $7,336 \times 0.81 = 59.4$ pounds of copper, and since the roasted ore contains 0.51

$$\frac{59.4}{0.51} = 116.67 \text{ pounds of ore, the weight of ore used, on}$$

this basis, would be $59.4 \div 0.0041 = 14,493$ pounds. As for nickel, the pig-metal contains $7,336 \times 0.0394 = 289$ pounds of

$$\frac{289}{0.284} = 1,017.6 \text{ pounds of ore, the weight of ore used, on}$$

per cent, the weight of this used should have been $289 \div 0.0223 = 12,960$ pounds. These figures differ so much that we will make the calculation on the basis of the iron. There is iron present as follows:

	Pounds.
In the pig iron $7,336 \times 0.8690$	$= 6,375$
In the slag $5,062 \times 0.0084 \times 56/72$	$= 33$
Total in products	$= 6,408$

In 100 pounds of ore $65.43 \times 0.7 = 45.80$
 In 21 pounds limestone $21. \times 0.0060 \times 0.7 = .08$
 In 27.5 pounds of charcoal $27.50 \times 0.0025 \times 0.7 = .04$

In charge, per 100 pounds of ore used = 45.92
 Therefore, ore necessary to supply the iron in products:

Pounds.

$6.408 \div 0.459 = 13.961$
 with which will be used:
 Charcoal = $13.961 \times 110/400 = 3.839$
 Limestone = $13.961 \times 85/400 = 2.967$

We are now ready to construct the balance sheet as soon as we assume probable values for the composition of the volatile matter and ash of the charcoal. The ash might be taken as containing on an average: K²O 15 per cent, CaO 40, MgO 20, MnO 15, and Fe²O³ 10 per cent. The volatile matter is due to insufficient charring, and the gases given off on heating may be assumed as, by volume, CO² 25 per cent, CO 15, H² 50, CH⁴ 10. This would make the volatile matter to contain, in per cents by weight, carbon 33.7, oxygen 58.5, hydrogen 7.8 per cent. (The verification of this last statement is a nice little exercise in chemical arithmetic.) The full statement of the elementary composition of the charcoal, for use in making the balance sheet, is therefore:

	Per Cent.
Fixed carbon.....	55.90
Volatile carbon.....	9.46
Volatile hydrogen.....	2.19
Volatile oxygen.....	16.43
Moisture.....	13.48
K ² O.....	0.38
CaO.....	1.02
MgO.....	0.51
MnO.....	0.38
Fe ² O ³	0.25

Balance Sheet, per 7,336 Lbs. Pig-metal.

Charges.	Pig-metal.	Slag.	Gases.
Roasted Ore..... 13,961			
Fe ² O ³ 9,135	Fe..... 6,375	FeO... 25	O..... 2,735
NiO..... 397	Ni..... 289	NiO... 28	O..... 80
CuO..... 71	Cu..... 59		O..... 12
SiO ² 1,530	Si..... 384	SiO ² ... 1,091	O..... 55
Al ² O ³ 461		Al ² O ³ ... 461	
CaO..... 547		CaO... 166	O..... 109
MgO..... 493		MgO... 493	
SO ² 545	S..... 1	CaS... 490	O..... 327
P ² O ⁵ 4	P..... 4		O..... 0
H ² O..... 778			H ² O... 778
Limestone..... 2,967			
CaO..... 1,543		CaO... 1,542	O..... 0
MgO..... 62		MgO... 62	
Fe ² O ³ 18		FeO... 16	O..... 2
Al ² O ³ 6		Al ² O ³ ... 6	
SiO ² 51		SiO ² ... 51	
P ² O ⁵ 0		P ² O ⁵ ... 0	
SO ² 4		CaS... 4	
CO ² 1,280			CO ₂ ... 1,280
Charcoal dust... 3,839			
Fixed C..... 2,146	C..... 224		C..... 1,922
Vol. C..... 363			C..... 363
Vol. H..... 84			H..... 84
Vol. O..... 631			O..... 631
H ² O..... 518			H ₂ O... 518
K ² O..... 15		K ² O... 15	
CaO..... 39		CaO... 39	
MgO..... 20		MgO... 20	
MnO..... 15		MnO... 15	
Fe ² O ³ 10		FeO... 9	O..... 1
Electrode..... 66			
C..... 66			C..... 66
20,828	7,336	4,533	8,962

There is a lack of close correspondence between the weights and compositions of slag, as observed and as calculated, due evidently to inaccurate sampling and analyses of the roasted ore and slag.

(2) From the balance sheet we can deduce the heat evolved and absorbed in the chemical reactions in the furnace. The more involved items are calculated as follows:

Oxidation of carbon to CO: All the carbon put into the furnace as fixed carbon goes out as either CO or CO², except that going into the pig iron. The carbon burnt to CO in the furnace may be, therefore, taken as $1,922 + 66 = 1,988$ pounds, evolving $1,988 \times 2,430 = 4,830,800$ pound-Calories.

Oxidation of CO to CO²: There is given up in the furnace, by the reductions accomplished, 3,021 pounds of oxygen, of which $1,988 \times 16/12 = 2,651$ pounds would burn fixed carbon to CO, as above shown, leaving 370 pounds to burn CO to CO². This would oxidize $370 \times 28/16 = 648$ pounds of CO to CO², which would evolve $648 \times 2,430 = 1,574,600$ pound-Calories.

Heat energy of electric current: One kilowatt-second is 0.239 kilogram-Calories, or 0.527 pound-Calories. The current being on 56 hours 20 minutes, or 202,800 seconds, the heat equivalent of the current used is:

$$0.527 \times 202,800 \times 165.65 = 17,704,000 \text{ pound-Calories.}$$

Heat in escaping gases: We are here confronted with the fact that no observation of the temperature of these was given. There is no essential reason why they should escape very hot from the furnace, if properly run and conducted, so we will assume a maximum temperature of 500° C. The gases would consist of $1,958 + 2,651 - 648 = 4,061$ pounds of CO and $648 + 370 = 1,018$ pounds of CO², from the oxidation of fixed carbon in the furnace; plus 1,280 of CO² from the limestone, and 666 pounds CO², 254 pounds CO, 97 pounds CH⁴ and 60 pounds of H² from the volatile matter of the charcoal. To these must be added 1,290 pounds of water vapor. The heat is, therefore:

CO	4,315 lbs. =	54,800 ft ³ × 0.304 =	16,650 oz. Cal.
CO ²	2,964 lbs. =	23,950 ft ³ × 0.480 =	11,500 "
CH ⁴	97 lbs. =	2,150 ft ³ × 0.490 =	1,050 "
H ²	60 lbs. =	10,700 ft ³ × 0.304 =	3,250 "
H ² O	1,290 lbs. =	25,500 ft ³ × 0.415 =	10,550 "
Sum	117,100 ft ³		43,000 " per 1°
			= 21,500,000 " per 500°
			= 1,343,750 lb.-Cal.

The other items of the heat balance sheet are almost self-explanatory, and the complete balance is as follows:

Heat Available.

	Pounds-Cal.
Energy of the electric current	= 17,704,000
Oxidation of C to CO	= 4,830,800
Oxidation of CO to CO ²	= 1,574,600
Combination of C with Fe ² 224 × 705	= 157,900
Combination of Ca with S 220 × 2,947	= 648,300
Formation of silico-aluminate slag (SiO ² + Al ² O ³) 1,609 × 150	= 241,400
Total	= 25,157,000

Heat Distribution.

Reductions:	
Fe ² O ³ to Fe	$6,375 \times 1,746 = 11,130,750$
NiO to Ni	$289 \times 1,051 = 303,750$
CuO to Cu	$59 \times 593 = 35,000$
SiO ² to Si	$384 \times 7,000 = 2,688,000$
SO ² to S	$218 \times 2,872 = 626,100$
P ² O ⁵ to P	$4 \times 5,892 = 23,550$
CaO to Ca	$274 \times 3,288 = 900,900$
Fe ² O ³ to FeO	$50 \times 446 = 22,300$
Expulsion of CO ² from flux:	$1,280 \times 1,026 = 1,313,300$

Evaporation of H ₂ O	$1,290 \times 606.5 =$	783,400
Sensible heat in gases		$= 1,343,750$
Sensible heat in pig iron	$7,336 \times 400 =$	2,934,400
Heat in slag	$4,533 \times 600 =$	2,719,800
Loss by radiation, conduction, etc		$= 332,000$

Total = 25,157,000 (2)

(3) The essential work done by the furnace is the reductions, evaporation of moisture and decomposition of carbonates. The heat in slag, iron, gases and radiation loss are all susceptible of diminution or of being more or less returnable to the furnace. The usefully applied heat is, therefore, 17,827,050 pound-Calories. To produce this there was consumed the 25,157,000 pound-Calories actually generated, and there was wasted 13,390,000 pound-Calories, the calorific power of the gases escaping from the furnace, which should have been generated or might be utilized, making a total of 38,527,000 pound-Calories disposable.

The working thermal efficiency over all was therefore:

$$\frac{17,827,050}{38,527,000} = 0.46 = 46 \text{ per cent.} \quad (3)$$

PRODUCTION OF STEEL.

There are three methods of producing steel electrically which are practicable. First, the electric furnace may replace the crucible simply as a melting apparatus, in producing a cast steel from cemented bars; second, the electric furnace may replace the crucible or open-hearth furnace as an apparatus in which to melt together wrought iron and pure cast iron, such as washed pig metal, although in this operation the electric furnace is more like the crucible method, in that there is not necessarily any oxidation of the metal or of its impurities in the operation; third, the electric furnace may be used to melt or keep melted cast iron, while its impurities are oxidized out by additions of iron ore, in this operation resembling the Uchatius method of making crucible steel or the "pig and ore" process of making steel in the open-hearth furnace.

The particular advantages possessed by the electric furnace processes are, compared with the crucible process, the larger quantities in which the steel can be made in one operation, the absence of carbon in the furnace lining, thus controlling better the carbon and silicon in the steel, and the higher temperature, enabling a more basic slag to be kept fluid and thus sulphur to be better eliminated; the advantages compared with the open-hearth furnace are the absence of gases of combustion in contact with the metal, and the higher temperature available, which permits of very basic, refractory slags being made and kept thinly fluid, and thus gives better control of sulphur and phosphorus. In addition to these, in both cases, may be mentioned the commercial advantages, for the saving in crucibles alone makes the electric furnace superior in this respect to the crucible process, and the electric furnace can compete successfully as regards cost with the regenerative open-hearth furnace wherever water power costs less than \$10.00 per horse-power-year and coal costs over \$5.00 per ton.

A particular point to be noted is, that when heating by combustion is used the efficiency of the absorption of heat by the charges decreases very rapidly as the temperature gets higher. For instance, if a cold ingot of iron is placed in a furnace the temperature of which is 1,500°, the iron absorbs heat with very great rapidity from the start up to, say, 1,000°, but with decreasing rapidity thereafter. The rate of transfer of heat from the gases to the iron is proportional to the difference of temperature, and is some fifteen times as fast when the iron is at 0° as when it is at 1,400°. If the efficiency of the heating by furnace gases is, say, 25 per cent in bringing metal up to 1,500°, it is likely that the distribution of this efficiency is distributed about as follows:

Heating from 0° to 500°	45 per cent efficiency
Heating from 500° to 1,000°	27 " "
Heating from 1,000° to 1,500°	3 " "

On the other hand, the conversion of electrical energy into heat in the substance of the material to be heated is not a contact or transfer phenomenon, but a thermodynamically frictionless transfer of 100 per cent efficiency, and equally so at the highest as at the lowest temperatures. Only radiation and conduction losses need be considered, the problem is not how much heat can you get into a body but how much can you keep in; it is already all in, 100 per cent of it, to start with. In a large, properly designed electric furnace the radiation and conduction losses of heat, even working to the highest temperatures, can be kept at 15 to 25 per cent of the total heat generated, giving an efficiency of 75 to 85 per cent.

It may very well be, that there are places where the relative prices of coal and electric power are such that coal is the cheaper for heating to 500°, at 45 per cent efficiency, or to 1,000° at 36 per cent efficiency, or even to 1,500°, at 25 per cent efficiency, but that the combination of heating by fuel to 1,000° at 36 per cent efficiency and then by electricity from 1,000° to 1,500°, at 70 per cent efficiency, would be the cheaper plan, or even by fuel to 500°, at 45 per cent efficiency, and then by electricity from 500° to 1,500°, at, say, 75 per cent efficiency, would be commercially advantageous.

Illustration: Steel bars are to be melted in an electrical furnace. It takes 300 Calories effective in the steel per kilogram to heat it to a tapping heat; the electric furnace supplies this at a net thermal efficiency of 75 per cent. To heat the bars to 750°, cherry red, without melting them, requires 88 Calories, or 29 per cent of the total. If the bars were heated in a coal furnace to 750°, and then transferred to the electric furnace, some 25 per cent of the electrical power might be saved. If this heating were done by coal having a calorific power of 8,500, at a thermal efficiency of 25 per cent, there would be needed 40 grams of coal. The question is, therefore, the relative cost of $88 \div 0.75 = 117$ Calories delivered electrically and 40 grams of coal. The former requires $117 \div 0.239 = 490$ kilowatt-seconds = 8.2 kilowatt-minutes = 0.14 kilowatt-hours. At \$10.00 per kilowatt-year (8,760 hours) this would cost $0.14 \times 0.114 = 0.016$ cents. At \$5.00 per metric ton the coal would cost $0.040 \times 0.5 = 0.020$ cent. Under such assumed conditions of cost of power and of coal the electrical heating, even up to 750°, would be the cheaper.

Problem 74

In an induction electric furnace of 170 kilowatts capacity, 4.7 tons of steel is made per day by melting together cold-washed pig iron and scrap iron, the melted steel carrying 350 Calories per kilogram.

Required:

(1) The electric energy in kilowatt hours required per ton of steel produced.

(2) The thermal efficiency of the furnace.

(3) If one-third the material used were put into the furnace melted, carrying 275 Calories per kilogram, what would be the production per day and the power required per ton of steel?

Solution:

(1) Energy for 4.7 tons	=	170 kw-days.
Energy for 1.0 ton	=	36.2 "
Energy for 1.0 ton	=	0.10 kw-year
Energy for 1.0 ton	=	869 kw-hours.

(1)

(2) 1 kw-hour = $0.239 \times 60 \times 60 =$	860 Calories.
869 kw-hours	= 747,340 "
Heat in 1 ton of steel = $350 \times 1,000 =$	350,000 "

$$\text{Thermal efficiency} = \frac{350,000}{747,340} = 0.47 = 47 \text{ per cent.}$$

(3) Heat in melted material used per kilogram of steel produced = $275 \times 1/3 = 92$ Calories.

Heat to be supplied by the current $350 - 92 = 258$ Calories.
Production per day under these conditions:

$$4.7 \times \frac{350}{258} = 6.4 \text{ tons.} \quad (3)$$

Relative times for the heats = 1 to 0.74.

Energy required per ton = $170 \div 6.4 = 26.6$ kw-days.
 $= 0.07$ kw-year.
 $= 638$ kw-hours. (3)

Problem 75.

An electric steel furnace running at full heat, and containing about 2,500 kg. of steel, loses by radiation, etc., 250,000 Calories per hour; 2,500 kg. of melted pig iron is run into the hot furnace, carrying 250 Calories per kilogram, and it is treated with 500 kg. of iron ore, previously heated to 500° C., and 50 kg. of limestone added cold. The steel produced carries 400 Calories per kilogram and the slag 600 Calories. The operation lasts 1 hour. Assume the following composition of materials used and made:

Pig Iron.	Iron Ore.	Limestone.	Steel.
Fe.....96.656	Fe ₂ O ₃ ...85.93	CaO.....53.74	Fe.....99.60
C.....2.700	FeO.....3.96	MgO.....0.17	C.....0.11
Si.....0.600	SiO ₂5.50	SiO ₂3.14	Si.....0.11
Mn.....0.025	MnO.....0.63	Fe ₂ O ₃ ...0.18	Mn.....0.15
S.....0.007	Al ₂ O ₃ ...0.76	Al ₂ O ₃ ...0.32	S.....0.02
P.....0.012	CaO.....2.23	P ₂ O ₅ ...0.006	P.....0.01
	MgO.....0.97	S.....0.001	

The bath was treated by the final addition of 10 kg. of cold ferro-manganese, carrying 80 manganese, 16 iron and 4 carbon. The steel obtained weighed 2,630 kg.

Required:

(1) A balance sheet of materials entering and leaving the furnace.

(2) The weight and percentage composition of the slag.

(3) A balance sheet of the heat received and distributed.

(4) The net power required to run the furnace and the cost of power per ton of steel made, at \$25.00 per kilowatt-year.

Balance Sheet.

Charges.	Steel.	Slag.	Gases.
Pig iron.....(2500 Kg.)			
Fe.....2416.4	2416.4		
C.....67.5	2.5		
Si.....15.0	2.9	SiO ₂ ...24.9	
Mn.....0.6		MnO...0.8	
S.....0.2	0.5		
P.....0.3	0.3		
Ore.....(500 Kg.)			
Fe ₂ O ₃ ...429.7	203.1	FeO...125.6	O.....87.2
FeO.....19.8		FeO...19.8	
SiO ₂ ...27.5		SiO ₂ ...27.5	
MnO.....3.2		MnO...3.2	
Al ₂ O ₃ ...3.8		Al ₂ O ₃ ...3.8	
CaO.....11.2		CaO...11.2	
MgO.....4.8		MgO...4.8	
Limestone.....(50 Kg.)			
CaO.....26.9		CaO...26.9	
MgO.....0.1		MgO...0.1	
SiO ₂ ...1.6		SiO ₂ ...1.6	
Fe ₂ O ₃ ...0.1		FeO...0.1	
Al ₂ O ₃ ...0.2		Al ₂ O ₃ ...0.2	
CO ₂ ...21.2			CO ₂ ...21.2
Ferro-manganese (10 Kg.)			
Fe.....1.6	1.6		
C.....0.4	0.4		
Mn.....8.0	3.9	MnO...5.3	
	3060.0	2631.6	255.8
			173.4

(2) The slag contains:

	Per Cent.
SiO ₂	54.0 pounds = 21.1
Al ₂ O ₃	4.0 " = 1.5
CaO.....	38.1 " = 14.9
MgO.....	4.9 " = 1.9
FeO.....	145.5 " = 56.9
MnO.....	9.3 " = 3.6
	255.8 " = 99.9 (2)

Heat Available.

Electric current:	Calories.
Oxidation of C to CO	65.0 × 2,430 = 157,950
Oxidation of CO to CO ₂	0.9 × 2,430 = 2,190
Oxidation of Si to SiO ₂	12.1 × 7,000 = 84,700
Oxidation of Mn to MnO	4.7 × 1,653 = 7,770
Formation of slag	22.6 × 150 = 3,390
Heat in melted pig iron	2,500 × 250 = 625,000
Heat in iron ore	500 × 77 = 38,500
Sum = x +	919,500

Heat Distribution.

	Calories.
Heat in melted steel	2,630 × 400 = 1,052,000
Heat in melted slag	256 × 600 = 153,600
Reduction of Fe ₂ O ₃ to FeO	386.7 × 446 = 172,470
Reduction of FeO to Fe	203.1 × 1,173 = 238,240
Separation of carbon from iron	65 × 705 = 45,800
Heat in gas at 1,500°:	
CO = 151.7 kg. = 120.4 m ³ × 0.32 × 500 =	19,260
CO ₂ = 1.4 kg. = 0.7 m ³ × 0.60 × 500 =	200
Loss by radiation, etc.	= 250,000

Total = 1,931,570

Heat to be supplied by current:

$$x = 1,931,570 - 919,500 = 1,012,070 \text{ Calories.} \quad (3)$$

(4) One kilowatt furnishes per hour 860 Calories, therefore, the power required to run the furnace is:

$$1,012,070 \div 860 = 1,177 \text{ kilowatts.} \quad (4)$$

At \$25.00 per kilowatt-year a kilowatt-hour would cost

$$\$25.00 \div 8,760 = 0.2854 \text{ cents.}$$

And the power to run the furnace 1 hour would cost

$$0.002854 \times 1,177 = \$3.36.$$

And the cost per ton of steel:

$$\$3.36 \div 2.630 = \$1.32. \quad (4)$$

Problem 76.

It is desired to design a plant for the electro-deposition of pure iron by the Burgess process (see Transactions American Electrochemical Society, Vol. V., p. 201). The desiderata and data are as follows:

Output, 25 metric tons per day.

Current density, 110 amps. per square meter.

Anodes, 0.75 × 0.5 meters immersed × 3 m.m. thick.

Cathodes, 0.75 × 0.5 meters immersed × 1 m.m. thick at starting.

Cathodes to be run until deposit is 1.5 c.m. thick on each side.

Anodes run until 0.9 consumed.

Tanks, 1.00 meter deep, 0.6 m. wide, 2 m. long inside, filled to within 0.10 meter of top with electrolyte.

Working distance between anode and cathode 6 centimeters at starting.

Electrolyte contains 10 per cent FeSO₄, 7H₂O, and 5 per cent (NH₄)₂SO₄, specific gravity 1.1, electrical resistivity 20 ohms per centimeter cube.

Voltage drop in connections and conducting rods 0.3 volt per tank.

Main conductors carry 2 amps. per m.m. square of section.

Net cost of electrical power, \$25.00 per kilowatt-year.

Requirements:

(1) The number of anodes and cathodes per tank and the number of tanks in the plant and their arrangement.

(2) The weight of anodes and cathode sheets, increasing the weight of immersed part 10 per cent. Specific gravity of the rolled iron 7.9.

(3) The weight of ferrous sulphate and ammonium sulphate required to start the plant.

(4) The drop of potential across the electrodes at starting

and at the close of a deposition; the drop of potential per tank; the total voltage needed at starting the plant and when it is in regular operation.

(5) The cross sectional area of the main conductors.

(6) The time required to consume an anode plate, i. e., in dissolving iron away equal to 0.9 of its weight.

(7) The time required to deposit a full cathode plate; specific gravity of the deposit 7.6.

(8) The electric power required to run the plant and its cost per ton of iron deposited.

Solution:

(1) 110 amps. per square meter deposits per day:

$$0.00001036 \times 110 \times \frac{56}{2} \times 60 \times 60 \times 24 = 2,757 \text{ grams Fe.}$$

Therefore, depositing surface required:

$$25,000 \div 2,757 = 9,140 \text{ square meters.}$$

Since one cathode plate has a depositing area on both sides of $0.75 \times 0.5 \times 2 = 0.75$ square meter, the number of cathode plates required in the whole plant is:

$$9,140 \div 0.75 = 12,187.$$

In one tank, if there are x cathodes and $x + 1$ anodes, the thickness of these plates at starting is, in millimeters, $x + 3$ ($x + 1$) = $4x + 3$ m.m. The number of spaces between anodes and cathodes being $2x$, and each of these being 6 c.m. = 60 m.m. at starting, the spaces are $120x$ m.m. The length of the tank being, inside, 2,000 m.m.:

$$124x + 3 = 2,000$$

whence

$$x = 16.1.$$

Each tank will therefore contain 16 cathodes and 17 anodes, at a distance apart, at starting of

$$\frac{2,000 - 16 - 3(17)}{2(16)} = 60.4 \text{ m.m.} \\ = 6.04 \text{ c.m.} \quad (1)$$

Since we need 12,187 cathode plates we need

$$12,187 \div 16 = 761.7 \text{ tanks.}$$

Which means that we would use 762 tanks. (1)

The arrangement of the tanks in series and groups of series can be best discussed when we know the voltage drop per tank, grouping them so as to absorb either 110 or 220 volts per series in one group.

(2) An anode sheet weighs:

$$75 \times 50 \times 0.3 \times 7.9 \times 1.1 = 9,776 \text{ grams,}$$

of which there is immersed 8,888 grams.

The 17 anodes per tank weigh altogether:

$$9,776 \times 17 = 166.2 \text{ kg.,}$$

and the 16 cathodes, which are one-third as thick:

$$3,259 \times 16 = 52.1 \text{ kg.}$$

In the whole plant, at starting, the weights will be:

$$\begin{array}{ll} \text{Anode sheets} & 166.2 \times 762 = 126,544 \text{ kg.} \\ \text{Cathode sheets} & 52.1 \times 762 = 39,700 \text{ "} \end{array} \quad (2)$$

(3) Volume of liquid in tank:

$$(1 - 0.1) \times 0.6 \times 2 = 1.08 \text{ cubic meters.}$$

Weight of solution per tank:

$$1.08 \times 1,000 \times 1.1 = 1,188 \text{ kg.}$$

Weight of dissolved salts:

$$\text{Copperas} = 118.8 \text{ "}$$

$$\text{Ammonium sulphate} = 59.4 \text{ "}$$

Weight in the whole plant:

$$\text{Copperas} = 90.5 \text{ tons.}$$

$$\text{Ammonium sulphate} = 45.3 \text{ "} \quad (3)$$

(4) At starting the surface of the electrodes are 6.04 c.m. apart, and 110 amps. passes through each square meter of electrode surface; therefore, $110 \times 0.375 = 41.25$ amps. pass from each free side of each anode plate to the corresponding side of a cathode plate. Neglecting the small cross sectional area of electrolyte outside the plates, the resistance of each space would be

$$\frac{20 \times 6.04}{75 \times 50} = 0.0322 \text{ ohms,}$$

and the drop of voltage across two electrodes:

$$0.0322 \times 41.25 = 1.33 \text{ volts.}$$

If we take into account the 5 c.m. free space at the sides of each electrode, and allow an equal amount as effective beneath, the cross-section of the electrolyte may be taken as

$$(75 + 5) \times (50 + 10) = 4,800 \text{ c.m.}^2,$$

and the resistance between two plates:

$$20 \times 6.04 \div 4,800 = 0.0252 \text{ ohms,}$$

and the drop $0.0252 \times 41.25 = 1.04$ volts.

This value is the more probable one of the two.

At the close of a deposition, neglecting the decreased thickness of the thin anode plates, the working distance is decreased by $1.5 \times 2 = 3$ centimeters, and the voltage drop between plates will then be:

$$\frac{20 \times 3.04}{4,800} \times 41.25 = 0.52 \text{ volt.}$$

In both cases the voltage drop in contacts and conductors being 0.3 volt, the working voltage per tank will be:

	Volts.
At starting.....	1.34
At end.....	0.82
Average	1.08

The voltage needed at the generators can only be calculated when we assume a plan of grouping the 762 tanks. If we assume 110 volts to be desired at the generators, we could run 102 tanks in one series, which would give $7\frac{1}{2}$ series. If we used 220 volts at the generators 2 series of 190 cells and 2 of 191 would absorb at starting 255 and 256 volts respectively, but when in regular running, with tanks in all stages of deposition, 205 and 206 volts. This would be a reasonable and practicable arrangement. In reality, at least one if not two additional tanks would be slipped into each series, for at least that number would be out of circuit continuously, being cleaned and made ready for re-starting.

(5) The amperes per tank would be:

$$0.75 \times 0.5 \times 2 \times 16 \times 110 = 1,320.$$

And the area of the main conductors in each series:

$$1,320 \div 2 = 660 \text{ sq. m.m.} \quad (5)$$

(6) The part of the anode sheet immersed weighs 8,888 grams, of which 0.9 is 8,000 grams, and if the anode is an intermediate one it is corroded on both sides, and receives, therefore, 85 amps. of current. This current dissolves, per second:

$$0.00001036 \times 28 \times 85 = 0.024657 \text{ grams.}$$

And therefore the anode sheet will last

$$\begin{array}{l} 8,000 \times 0.024657 = 324,000 \text{ seconds.} \\ = 90 \text{ hours.} \end{array} \quad (6)$$

(7) The weight of deposit on both sides of a cathode plate is

$$75 \times 50 \times 2 \times 1.5 \times 7.6 = 85,500 \text{ grams.}$$

And the time required to deposit this, since it is deposited by 85 amps., is

$$\begin{array}{l} 85,500 \div 0.024657 = 3,467,600 \text{ seconds.} \\ = 40 \text{ days 3 hours.} \end{array} \quad (8)$$

(8) Each series requires 1,320 amps. at 205 volts, or

$$1,320 \times 205 \div 1,000 = 270.6 \text{ kilowatts.}$$

The three series therefore require 812 kilowatts, which will cost

$$\$25.00 \times 812 \div 365.25 = \$55.68 \text{ per day.}$$

An average cost of power per ton of iron refined of

$$\$55.68 \div 25 = \$2.23. \quad (8)$$

The other items of cost in a well conducted refinery will about equal this sum, making the total cost of refining about \$4.50 per ton of pure iron. With cheap soft steel used as the raw material, there is a striking possibility of such a process

being commercially practicable for furnishing one of the raw materials for producing the finest qualities of steel, the other raw material being washed pig metal of standard quality. We commend this possibility to the attention of the makers of fine steel.

The Aussig Glocken Process for Alkali Chloride Electrolysis.

BY DR. OTTO STEINER.

Electrochemical methods are generally employed in chemical industries for the sake of simplifying the operation. Purely chemical processes, like the LeBlanc, Weldon, Deacon and Solvay processes, involve a whole series of different reactions in order to get the desired end products from common salt. The electric current, on the other hand, decomposes the salt in one single apparatus and in one single operation. The immediate products of electrolysis are chlorine gas at the anode and caustic soda at the cathode. It is only necessary to so construct the apparatus as to prevent the reunion of these two products.

For this purpose it is necessary to separate the anodic liquid, in which some chlorine gas is dissolved, from the cathode liquid which contains the caustic alkali. While the chlorine set free at the anode mostly escapes as gas, yet some of it dissolves in the liquid. If this anode liquid containing the chlorine then mixes with cathode liquid containing hydroxide hypochlorite will be formed.

By application of a diaphragm, it is possible to separate the anolyte from the catholyte so as to prevent the mechanical mixing of the two solutions, without preventing the passage of the electric current.

But the action between the two solutions is due not only to mechanical mixing, but is also electrochemical in nature, since the caustic alkali in the catholyte participates in the conduction of the current. The OH ions of the catholyte will travel through the diaphragm into the anolyte. In this way caustic alkali is transported towards the anode in spite of the diaphragm. In consequence of this, it is impossible on a large commercial scale to manufacture highly concentrated caustic solutions by the diaphragm process without running the risk of greatly reducing the ampere-hour efficiency, together with other troublesome disadvantages.¹

Various methods have been suggested to prevent the electrochemical reaction between the two liquids. There are some hundred different processes, aiming at this end, but only those of special simplicity have proven industrially successful. This is especially the case for the Aussig "Glocken" (bell) process of the "Oesterreichische Verein für Chemische und Metallurgische Production in Aussig" (Bohemia). This process is now used on a very large scale (about 6,000 h.p.) in Austria as well as in Germany.

The general principle of the process is shown in Fig. 1. An earthenware bell (Glocke) is suspended in the salt solution in such a way that it does not reach down to the bottom. The anode is inside of the bell, the cathode outside.

The anode liquid, which has a green color on account of its content of dissolved chlorine gas, is of lower specific gravity and fills the upper part of the bell. The colorless cathode liquid fills the part underneath as well as the whole outside of the bell.

¹ The formation of hypochlorite by reaction of the anode liquid with the cathode liquid can be theoretically explained as follows: There is always hydrolysis in the anode liquid $\text{Cl}_2 + \text{H}_2\text{O} \rightleftharpoons \text{H} \cdot \text{Cl}' + \text{H} \cdot \text{ClO}'$. The two acids react with the cathode liquid as follows: $\text{K} \cdot \text{OH}' + \text{H} \cdot \text{OH}' + \text{H} \cdot \text{ClO}' + \text{H} \cdot \text{Cl}' = 2\text{H}_2\text{O} + \text{K} \cdot \text{OCl}' + \text{K} \cdot \text{Cl}'$. The KOCl and H_2O are very slightly dissociated, so that more hydrolysis will take place. Hypochlorite is again formed, and in that way the whole caustic alkali that enters into the anode liquid will be changed into hypochlorite.

Between both solutions is a layer of neutral solution, which remains at the same place, if the current, the content of hydroxide in the catholyte and the content of alkali chloride in the anolyte are kept constant. This can be accomplished in continuous operation by introducing the fresh salt solution into the bell above the anode, so as to distribute it very uniformly over the whole anode surface, and by continually removing the formed caustic solution from the cathode compartment by simply overflow.

The anolyte, containing chlorine, moves inside of the bell downwards in consequence of the continuous supply of fresh electrolyte. It finally passes outside of the bell, and the hydroxide of the catholyte will then react with the dissolved chlorine and form hypochlorite, which will be reduced to alkali chloride at the cathode. In practice this reaction between chlorine and hydroxide will cause a loss of 6 per cent in the ampere-hour efficiency.

The idea of permitting the anode liquid with the chlorine to

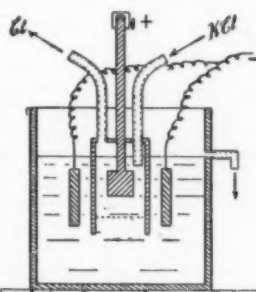


FIG. 1.—PRINCIPLE OF GLOCKEN PROCESS.

pass over directly to the cathode is rather audacious and is almost stunning at first sight, because this is exactly the thing which ought to be prevented—namely, the mixing of the anolyte with the catholyte. But as a strong salt solution does not dissolve more chlorine than 0.4 per cent of its weight, the loss

in ampere-hour efficiency is very small, it amounts on a large scale to only 6 per cent. On the other hand, this arrangement has other very great advantages. The liquid surrounding the anode is constantly maintained saturated with chloride and the OH ions, which migrate away from the cathode towards the anode, are thus prevented to reach the anode.

The reason is this: Both the OH ions and the Cl ions are migrating from the cathode to the anode, and the part which these two kinds of ions take in the electric conduction depends, according to Hittorf, on their number and their speed of migration. If, for example, the solution at the cathode contains 12 per cent KOH and 12 per cent KCl, it will be found that the negative ions traveling away from the cathode are almost completely OH ions.

But on their further advance towards the anode they will come into layers, which contain more KCl, and therefore more Cl ions, on account of the continuous supply of fresh salt solution at the anode. Therefore, the OH ions will gradually remain behind and the Cl ions will chiefly undertake the negative transmission of the current. At the anode itself the KCl concentration is maintained constantly at 20 per cent, so that the only negative ions moving at this place will be the Cl ions and no OH ions will reach the anode.

In expositions of the theory of the Glocken process the wrong idea is often found that the electrolyte moves inside of the bell downwards with the same speed with which the OH ions move upwards, and that it is necessary to regulate the rate of supply of fresh alkali-chloride solution in such a way that the neutral layer which forms between the anolyte containing chlorine and the caustic catholyte remains on the same place.

Even the German patent 141,187 of the Aussig Co. uses this explanation. It is said there: "2. The continuous admission of fresh electrolyte at the anode is so regulated, that the speed with which the anode liquid passes downward, is equal to that with which the caustic alkali moves upwards on account of ionic migration, so that the separating layer between anolyte and catholyte is maintained constantly at the same place." It

would probably be more exact to say that the separating layer is maintained at the same place by keeping the alkali chloride concentration constant in the surroundings of the anode.

For, in reality, the explanation of the patent is not admissible. The position of the neutral layer depends chiefly on the concentration of caustic alkali at the cathode and on the KCl concentration at the anode and not on the speed with which the solution moves. The speed with which the electrolyte inside of the bell moves downward (in practice 1 c.m. per hour) is almost insignificant, since the OH ions travel with a speed of 6 c.m. per hour upwards.

According to the patent these two speeds should be equal. But that is impossible in practice. To bring the downward speed of the solution up to 6 c.m. per hour it would either be necessary to accelerate the supply of fresh electrolyte so much that with a current density of 2 amperes per square decimeter of horizontal cross section of the bell (which is the limit on account of the disintegration of the anode material) a very weak caustic alkali solution would be obtained. Or it would be necessary to make the cross section of the bell below the anode so small that the voltage, and therefore the expenses for electric energy would be considerably increased. But all these things are unnecessary. If the liquid around the anode is continuously kept saturated with KCl or NaCl by careful subdivision and mixing of the fresh electrolyte, which is introduced into the bell, it is possible to prevent the OH ions from reaching the anode, provided there is no disturbing circulation of liquid within the bell.

For other reasons it would also be evidently impossible to regulate the admission of electrolyte so that the speed with which the anode liquid passes downwards is equal to that with which the OH ions move upwards. On that theory the neutral

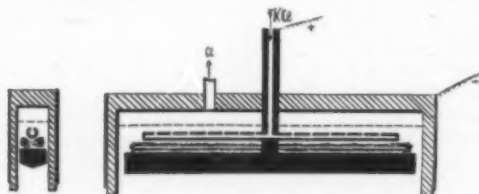


FIG. 2.—LATER CONSTRUCTION OF CELL.

separating layer would then only be stationary in the one case that both speeds are exactly equal. This can, of course, never be accomplished on a commercial scale, so that whenever the two speeds are not absolutely equal, the neutral layer would travel either upwards or downwards.

But, as a matter of fact, I have proved by experiments (*) that the neutral layer always remains at the same place, whatever the speed of the electrolyte may be, if only the current and the supply of fresh electrolyte are maintained constant. My explanation is therefore the only one admissible—namely, that the passage of the caustic alkali right to the anode is prevented in the Glocken process by keeping the liquid surrounding the anode as much as possible saturated with KCl, so that near the anode no OH ions participate in the negative transmission of the current.

The employment of a concentrated alkali chloride solution and a good distribution and mixing of the new with the old solution at the anode may also be used to advantage in the diaphragm and mercury processes where similar conditions exist.

The technical arrangement of the Aussig Glocken process requires a somewhat different construction, somewhat as shown in the adjoining Fig. 2. The even uniform distribution of the fresh solution over the whole anode surface and its thorough mixing with the solution, which already surrounds the anode, and which is weaker and therefore of lower specific

gravity, are of greatest importance. In the patents of the Aussig firm it is specified, that for this purpose the anode should be made so large that it fills almost completely the whole horizontal cross-section of the cell. Only a small space of a few millimeter is left between the anode and the cell wall, and in this narrow space the rising chlorine bubbles cause an intimate mixing of the solutions.

I have proved experiments, that this condition alone is not sufficient, but that it is also necessary that the upper surface of the anode, or at least the exterior rim of it, should be absolutely level and horizontal. If this surface is only a little inclined, all the fresh solution, being of higher specific gravity, will flow down in one stream ("schliere") from the lowest point of the anode surface without being distributed over the whole anode surface and without getting thoroughly mixed with all the old solution. As a result the anode liquid will get weaker in alkali chloride. OH ions will come near to the anode, oxygen will be set free, disintegration of the anode material will occur, in short, we have all troubles which it is of greatest importance to avoid.

Not only the flat form but also the absolutely horizontal position of the anodes is therefore of greatest importance and every bell must be an apparatus of precision if it shall operate properly. By careful observation of this precaution Acheson graphite will prove such an excellent anode material that it will last for five years.

To sum up the advantages of the Aussig bell process, first of all we have the simplicity of its apparatus, then the high ampere-hour efficiency, the high concentration of the caustic alkali produced and the long life of the anodes. Its disadvantage is that for a large scale production a very great number of small apparatus are necessary, since the bells cannot be made too large. The diaphragm and mercury processes have also their advantages and disadvantages and it depends on local conditions which process one shall select. With respect to simplicity the Aussig Glocken process will not be excelled by any other electrolytic alkali-chloride process.

The "Oesterreichischer Verein für Chemische und Metallurgische Production in Aussig" has not only invented the Glocken process but has also worked out all the details with so much success that the process is now used not only in Austria but in several places in Germany, in successful competition with the other older electrolytic processes.

The experiments on which the results given in this article are based were carried out under the direction of Prof. Dieffenbach at the Electrochemical Laboratory of the Institute of Technology of Darmstadt. The author wishes to express here his thanks to Prof. Dieffenbach for the assistance given him during the investigation.

CREFELD, Germany.

A 24-Ton Induction Furnace for Steel Manufacture.

The adjoining six illustrations show a 736-kw tilting Kjellin induction furnace for steel manufacture, which has just been completed at the Roehling iron and steel works in Voelklingen, Germany. This is the result of the success attained at the same works with two smaller induction furnaces. As was noticed in the paper of Mr. Hermann Roehling, on page 92 of our March issue, these two smaller furnaces have a capacity of 50 to 60 kg. (110 to 130 pounds) and of 300 kg. (660 pounds) respectively. The new furnace has a capacity of 24 tons of steel, 15 tons being poured at the end of a run, the balance remaining in the furnace for the next run.

The picture at the left in the first row shows the single-phase alternating-current generator which supplies the primary of the furnace with current. The picture at the right gives a view of the machine room. It illustrates the intimate relation which

* Zeit. f. Elektrochemie, 1904, pp. 322 and 325; see also *Electrochemical Industry*, 1904, p. 243.

exists between steel manufacture in the electric furnace and cheap power generation in gas engines.

The two pictures in the next row are two full views of the electric furnace, first in course of construction, then after completion. As already mentioned, the capacity is 24 tons of steel,

The two pictures in the last row give a rear view and a top view of the furnace. They show mechanical details with respect to the tilting arrangement, etc. The general features of the construction of the furnace are the same as shown in the diagrams on page 93 of our March issue. For the photo-

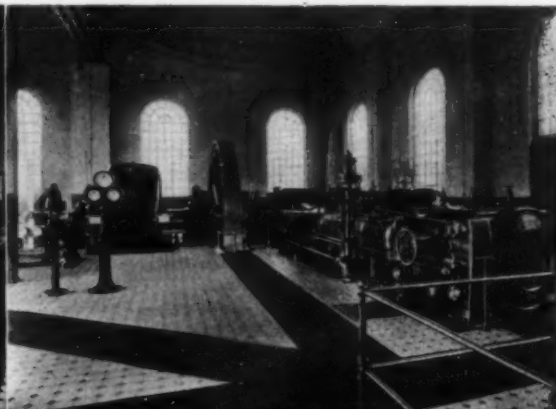
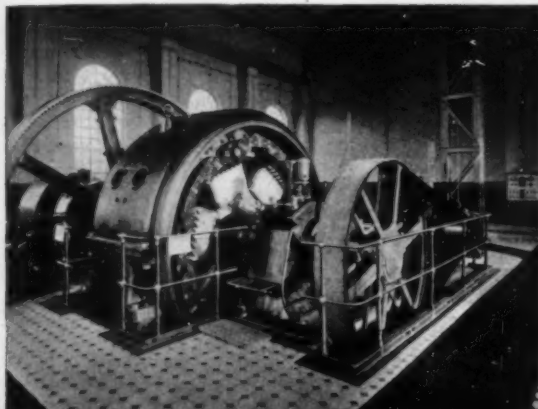


Fig. 1.—Alternator.

Fig. 2.—Furnace During Erection.

Fig. 3.—Top View of Furnace.

Fig. 4.—Generating Room.

Fig. 5.—Furnace After Completion.

Fig. 6.—Rear View of Furnace.

ELECTRIC STEEL FURNACE PLANT AT VOELKLINGEN.

15 tons being poured at the end of a run. The electric power for operating the furnace is 736 kilowatts, or approximately 1,000 electric horse-power. Many of our readers have certainly seen small experimental induction furnaces in operation, but these illustrations afford a novel sight. One is impressed by them with the fact that those German steel masters do not intend to use the electric furnace as a play-thing or a show-thing, but that they mean business.

graphs from which these pictures are reproduced we are obliged to the American Gröndal-Kjellin Co.

An interesting and quite novel feature of the same plant is a fourth induction furnace, now in course of construction. It has a capacity of 150 tons, but is not intended for steel making and refining proper, but will be used as a mixer and reservoir to keep the hot molten metal, which is tapped from the ordinary metallurgical furnaces and which is to be refined in the

electric furnace, in molten condition. From this reservoir molten metal is to be run into the electric furnace right after the completion of a run, so as to maintain as much as possible continuity of operation.

In this connection it is interesting to give a summary of electric induction furnaces installed for commercial operation. In this country we have a Colby induction furnace at the Diss-ton Steel Works in Philadelphia. In Europe the following Kjellin induction furnaces have been installed:

Gysingen, Sweden: 150 kw., 955 kg. (20,000 pounds). This is the original Kjellin furnace, reported on by the Canadian Commission (see our Vol I., p. 576, Vol. II, p. 479).

Gurtellen, Switzerland: International Calcium Co.: 320 kw.

Voelklingen, Germany: Roehling Works. Of the following four furnaces the first three are in operation, the fourth in course of construction: First, 50 to 75 kw., capacity 50 to 60 kg. (110 to 130 pounds); second, 110 to 120 kw., capacity 300 kg. (660 pounds); third, 736 kw., pour 15 tons (15,000 kg. or 33,000 pounds); fourth, a 150-ton mixer to keep the metal molten.

Essen, Germany: Krupp Works, 746 kw.

Sheffield, England: Vickers, Sons & Maxim, 200 kg.

Araga, Spain: 200 kw.

There are further in course of construction a 300 to 400-kw. furnace at Guldsmeshyttan, Sweden, and another furnace of the same capacity at Poldihütte.

From the above figures for the furnaces at Gysingen and Voelklingen the following interesting figures on the relation between capacity of furnace and required electric capacity are derived:

Capacity in kg. steel	50 to 60	300	955	15,000
Electric power in kw.	50 to 73	110 to 120	150	736
Approximate ratio of kw. to kg.	1	1-3	1-6	1-20

These figures are very instructive in showing in a general way how the electric power required per unit of output decreases with increasing capacity of the furnace. The electric energy, however, is only a single item in the cost sheet, and we have repeatedly pointed out that this item is comparatively small in the special case of manufacturing tool steels to compete with crucible steels.

To properly estimate the great progress which has been made by increasing the size of the furnace, it must be taken into account that other and really more significant items in the cost sheet are also substantially reduced, such as cost of labor per unit of output.

The chief point which we wish to make in this article is that large-size electric steel furnaces which enable a high economy of operation, are no longer only a possibility, but an established commercial fact.

Soap.

The New York Section of the Society of Chemical Industry held a meeting on April 10, the program for the evening being a symposium on soap. The following papers were on the program:

J. Lewkowitsch—Modern Views on the Construction of Soap.

W. C. Alpers—History and Uses of Soap in Pharmacy and Medicine.

Frank L. Randel—Raw Materials for Soap Making.

D. Wesson—Cotton Seed Soap Stock.

Martin H. Itmer—Laundry and Toilet Soaps.

J. Merritt Matthews—Textile Soaps.

J. F. Hinkley—The Recovery of Glycerine from Soap Lyes.

William Dreyfus—Liquid Soaps from a Sanitary Standpoint.

A. C. Langmuir—Analysis and Valuation of Crude Glycerine.

N. J. Lane—Determination of Castor Oil in Mixtures, Soaps and Alizarine Assistants.

The Use of Graphic Formulæ in Metallurgical Calculations.*

By DAVID H. BROWNE.

The daily records of furnace practice, the charge sheets, analyses, blast records, etc., are of value only in so far as they leave a permanent record in the minds of the furnace superintendent and his foremen, which record is termed "experience." As all these data, daily accumulating, become in time too numerous for the mind to carry, it is necessary to reduce them to terms of some common denominator and to group them in ways that will make them understood. In furnace practice also certain calculations are daily necessary, and these may be reduced to simple tables or charts.

The purpose of the present paper is to outline methods whereby the records of one particular ore may be tabulated and reduced to graphic formulæ. If a standard method, applicable to all ores, could be devised, then the science of metallurgy would approach the status of an exact science.

Suppose, for illustration, we have a large ore body consisting of chalcopyrite, pyrrhotite and diorite, and suppose that we have found by experience that the ore is regular in composition,

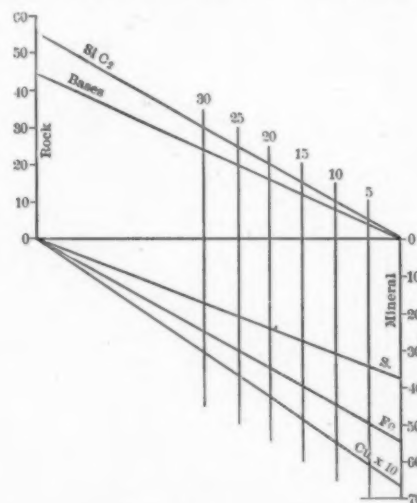


FIG. 1.—DIAGRAM FOR FINDING PERCENTAGE COMPOSITION.

varying only in the relative proportions of minerals and rock. A fair average analysis is, we will assume, as follows: 4.50 Cu, 40.06 Fe, 25.66 S, 18.20 SiO₂, 6.30 Al₂O₃, 3.00 CaO, 1.20 MgO.

We have first to reduce this to its mechanical composition, considering it as a mixture of chalcopyrite, pyrrhotite and rock. Starting with chalcopyrite we have the analysis and ratio of this mineral to guide us.

Chalcopyrite: Cu 34.5 per cent = 1,000 parts; S 35 per cent = 1,014 parts; Fe 30.5 per cent = 883 parts.

The ore has 4.5 per cent copper, therefore, to form chalcopyrite this copper must be combined with iron and sulphur in the following amounts: S = $4.5 \times 1.014 = 4.56$ part S and Fe = $4.5 \times .883 = 3.97$ parts Fe.

We have in the ore as chalcopyrite: Cu = 4.5 per cent, S = 4.56 per cent, and Fe = 3.97 per cent; total 13.03 per cent chalcopyrite.

Deducting this sulphur and iron from the amounts in the original analysis we have: S = $25.66 - 4.56 = 21.1$ sulphur left, and Fe = $40.06 - 3.97 = 36.09$ iron left.

Taking this 21.1 parts of sulphur left, we combine it with

* A paper presented at the Toronto meeting of the Canadian Mining Institute.

enough iron to form pyrrhotite, of which the analysis and ratio is: Fe 61 per cent = 1,564 parts, and S 39 per cent = 1,000 parts.

The iron combined with the remaining sulphur is: Fe = $21.1 \times 1.564 = 33.0$ per cent Fe as pyrrhotite. We subtract this from the iron we had left after the chalcopyrite was satisfied: $36.09 - 33.00 = 3.09$ iron left. This must be combined with oxygen, and as such will form almost 4 per cent iron oxide.

We can now reconstruct the mineral portion of the ore as follows:

	Cu	Fe	S
Chalcopyrite	4.5	3.97	4.56
Pyrrhotite		33.0	21.1
	4.5	36.97	25.66

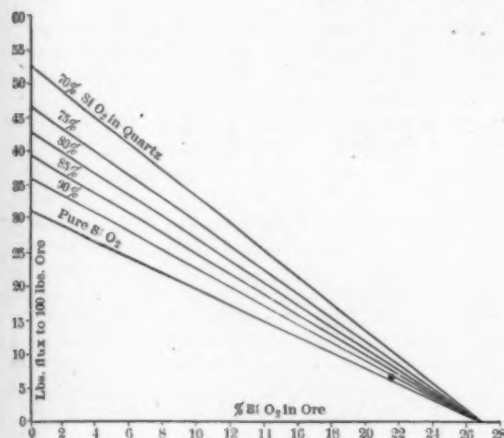


FIG. 2.—FLUX CHART.

Raising this from a total of 67.12 to 100 per cent we have 67 per cent Cu, 55.0 per cent Fe, 38.2 per cent S.

The rock materials which are left are:

	Per Cent.
FeO	4.00 = 12.2
SiO ₂	18.20 = 55.65
Al ₂ O ₃	6.30 = 19.26
CaO	3.00 = 9.1
MgO	1.20 = 3.6
	32.70 99.8

For metallurgical purposes we can group these constituents as 55.65 per cent silica and 44.2 per cent bases.

We have now the mechanical composition of the ore as 67.12 parts mineral, containing 6.7 per cent Cu, 55.0 Fe and 38.2 S, and 32.70 parts rock, containing 55.65 per cent SiO₂ and 44.2 bases.

We can now assume that any ore from this mine will, in large quantities, correspond to some definite mixture of this mineral and this rock, and we can plot a chart showing what the composition of any such mixture will be.

Lay off on quadrille paper a base line containing 100 horizontal divisions. At the left-hand end erect a perpendicular, which we will call "rock." Lay off on this rock line the distances from the base line corresponding to the percentage of silica and bases in "rock," and connect these points with the zero at the right-hand end of the base line. From this zero at the right-hand end of the base line drop a vertical, which we will call "mineral," and on this lay off with the same scale distances corresponding to the Cu, Fe and S in the mineral. Connect these points with zero at the left-hand end of the base line.

We have now, Fig. 1, a vertical cross-section of which at any point will give the percentage composition at this point. For

the sake of clearness we can lay out the copper contents on a scale ten times that of the other constituents.

Let us take silica as a standard, and lay off on Fig. 1 vertical lines corresponding to 5, 10, 15, 20, 25 and 30 per cent silica. We can now read off the ore composition for variations of 5 per cent silica.

For example: 20 per cent SiO₂, 15.5 bases, 24.5 sulphur, 35.5 iron, 4.30 copper.

We can now draw up a slag chart covering the grades of ore we most commonly use. We take the pure mineral and calculate the proper slag for it; then we calculate the slag for an ore containing 20 per cent silica. By plotting these and continuing the lines we obtain a slag chart for ore of any composition between pure mineral and pure rock.

The pure mineral contains 6.7 per cent Cu, 55.0 Fe and 38.2 S.

This sulphur will be roasted to 12 or 13 per cent before coming to the smelter bins. This does not alter the slag calculation. We decide to make a matte carrying 40 per cent copper and a slag carrying 33 per cent silica. It is well known that copper mattes with 79 per cent copper contain no iron, and that 64 per cent iron mattes carry no copper. So by plotting copper on a vertical scale and iron on a horizontal, and connecting 79 per cent Cu with 64 per cent Fe, we get a diagonal line showing the amount of iron in mattes of any copper content. From this we find that a 40 per cent copper matte carries 32 per cent iron.

We make, for the present, no deductions for slag loss, but calculate as follows: 6.7 parts copper forming a matte containing 40 per cent Cu and 32 per cent Fe will take 5.36 parts Fe into the matte. The mineral has 55 parts iron, and after deducting the iron which goes into the matte we have 49.64 parts iron to go into slag. One part iron makes 1.28 parts iron oxide, so 49.64 parts iron make 63.59 parts iron oxide. We have, therefore, 63.59 pounds iron oxide from 100 pounds

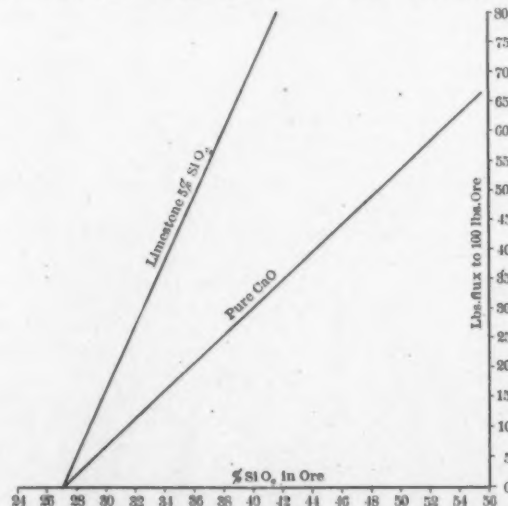


FIG. 3.—FLUX CHART FOR ORE WITH MORE THAN 27% SILICA.

mineral, and we have to add silica so that the silica is 33 per cent in the slag.

Let x = silica to be added, then $x = .33 (63.59 + x)$, and $x = 31.3$ pounds silica. Therefore, 100 pounds pure mineral requires 31.3 pounds silica. Take now the ore mixture with 20 per cent silica, which we found to contain: 20 per cent SiO₂, 24.5 S, 35.5 Fe, 15.5 other bases and 4.3 copper.

This 4.3 parts Cu passing into matte requires $4.3 \times 0.8 \text{ Fe} = 3.4$ pounds Fe. Deducting this from 35.5 Fe we have left 32.1 pounds Fe to be slagged. Calculating this to iron oxide, $32.1 \times 1.28 = 41.0$ pounds iron oxide to be slagged. The total in-

* Every 40 parts copper requires 32 parts iron to form matte, therefore 1 part copper requires 0.8 iron.

gredients passing into the slag are: 41.0 per cent iron oxide, 15.5 other bases, 20.0 silica (sum 76.5 per cent).

We have to add to this 20 pounds silica a certain amount x of outside silica, so that $20 + x$ will be 33 per cent of $76.5 + x$, hence $x = 8$ pounds silica.

We have thus found that a pure mineral requires 31.3 pounds silica and an ore with 20 per cent silica requires 8 pounds silica. We can plot these on a chart and read the amount of silica required for ore of any silica content. Such a chart is drawn in Fig. 2.

As we never have pure silica available for flux, we must lay off on the chart other lines corresponding to the amounts of quartz needed with varying silica content. Each 1 per cent of bases in the quartz abstracts 0.33 per cent of silica, so that a rock with 33 per cent silica would have no silica available. We can lay out a table showing the flux value of various grades of quartz.

Per Cent SiO_2 in Quartz.	Per Cent SiO_2 Available.	Lbs. Quartz to Equal 100 Lbs. Pure SiO_2
100	100	100
95	93.35	107
90	86.7	115
85	80.05	125
80	72.80	137
75	66.75	150
70	59.20	169
65	53.45	187
60	46.80	213

We can now plot on Fig. 2 the lines corresponding to the various percentage of silica in the quartz used for flux. The pure mineral takes 31.3 pounds of pure silica. If a quartz with 90 per cent silica is used, we have to take $31.3 \times 1.15 = 36$ pounds of this quartz. Making out a table of this we have: 100 pounds mineral requires 31.3 pounds silica; 36.0 pounds 90 per cent quartz; 39.1 pounds 85 per cent quartz; 42.8 pounds 80 per cent quartz; 46.9 pounds 75 per cent quartz; 52.8 pounds 70 per cent quartz.

A similar calculation can be made for ore containing 20 per cent silica: 100 pounds ore at 20 per cent silica requires 8 pounds silica; 9.2 pounds 90 per cent quartz; 10.0 pounds 85 per cent quartz; 10.9 pounds 80 per cent quartz; 12.0 pounds 75 per cent quartz; 13.5 pounds 70 per cent quartz.

Dotting in these points in their proper position and drawing the lines we complete Fig. 2.

We can now read off the amount of flux needed for any per cent silica in the ore mixture. For example: The ore is 13 per cent silica; the quartz the same day has 80 per cent silica. Finding 13 per cent silica on the base line we follow it up and find it intersects the 80 per cent silica line at 24 pounds. Therefore, 100 pounds of this ore will require 24 pounds of this quartz to make 40 per cent matte and 33 per cent slag.

These lines, it will be noted, all intersect at 27 per cent, showing that an ore with 27 per cent silica and with the other constituents as shown by Fig. 1, is self-fluxing for a 33 per cent silica slag. We may prolong this chart and see that ores with more than 27 per cent silica require the addition of a base, such as lime, or an iron ore, or a basic copper ore.

We will take limestone as an example, and assume we are using the same ore, making the same matte and slag, and using lime as the added base. Start the calculations with the pure rock which accompanies the mineral in our ore. This rock, by the calculation, contains 55.65 silica and 44.20 bases. We desire to make this into a slag with 33 per cent silica and 67 per cent bases. Evidently the amount x of added lime must be such that $44.2 + x = 67$ per cent of $100 + x$, hence $x = 66$ pounds CaO .

We can now make Fig. 3, laying off on the right-hand side a vertical on which we mark 66 pounds lime above the point showing 55.65 per cent pure silica. As 27 per cent silica ore is self-fluxing, we can draw a diagonal from this point to 66 pounds lime, and we have here a reading of the amount of

lime needed for all ore between 27 per cent silica and pure rock.

But as on the silica side we have no pure silica available, so on the lime side we have to work with crude limestone, and we have to make a table showing the flux value of these limestones. A pure silicate of lime with 33 per cent silica carries 67 per cent lime, or simplified, 1 pound silica equals 2 pounds of lime. So in a limestone with 5 per cent silica, this 5 per cent silica neutralizes 10 per cent lime and cuts down the fluxing power of the limestone. We make a table of this:

% Silica in Limestone.	% CaO in Limestone.	% Available CaO .	Lbs. Limestone to Equal 100 CaO .
0	56	56	178
5	53.2	43.2	231
10	50.4	30.4	328
15	47.6	17.6	568

Suppose in the present instance our limestone has 5 per cent silica, and as we have shown 100 pounds pure rock needs 66 pounds pure lime, we multiply 66 by 2.31 and find that 152.4 pounds of this limestone are required.

We can now complete Fig. 3, laying out the lines for pure lime and for any grade of limestone we have available.

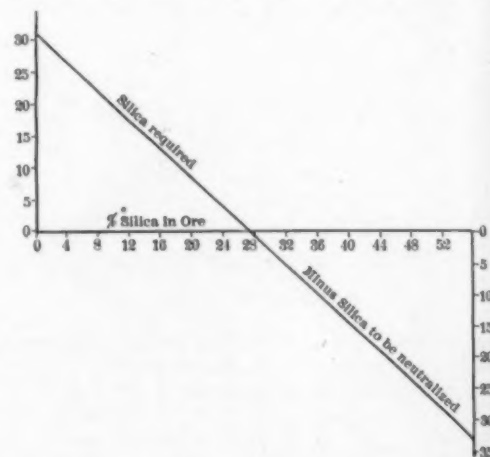


FIG. 4.—COMBINATION OF FIGS. 2 AND 3.

Suppose we have an ore with 38 per cent silica, Fig. 3 shows that 100 pounds such ore requires 60 pounds of our limestone. We can check this very readily. From Table 1 we find that this ore carries 38 per cent SiO_2 , 18 iron, 30 other bases, 12 sulphur and 2.0 copper. This copper requires 1.35 pounds iron in matte, which leaves $18 - 1.35 = 16.65$ pounds iron to go into slag. Multiplying 16.65 by 1.28 gives 21.3 pounds iron oxide to be slagged. The slag constituents are 38 per cent silica, 21.3 iron oxide, 30.0 other bases (sum, 89.3 per cent). Or adding together all the bases we have 51.3 per cent total bases and 38.0 per cent silica (sum, 89.3 per cent). We desire to make slag with 33 per cent silica and 67 per cent bases. Therefore $51.3 + x \text{ CaO} = .67 (89.3 + x)$ and $x = 25.8$.

Looking up Table 3, we find this checks out almost exactly. But our limestone has 5 per cent silica. Now, 5 per cent silica leaves 95 per cent CaCO_3 , or $95 \times .56 = 53.20$ CaO in this limestone. As 1 pound silica in the slag neutralizes 2 pounds bases, so 5 pounds silica in the limestone neutralizes 10 pounds CaO —so we have $53.20 - 10 = 43.20$ available CaO . We have

$$\text{to use } 25.8 \text{ pounds } \text{CaO}, \text{ therefore, we use } 25.8 \times \frac{100}{43.2} =$$

59.7 pounds of our limestone. Referring to Fig. 3 we find 60 pounds indicated, which proves the accuracy of our chart.

We could, however, have made this calculation in a much simpler way. From Fig. 2 we saw that the pure silica line ran

from 31.3 pounds required for pure mineral down to none for ore with 27 per cent silica. If we prolong this silica line diagonally to the right below our base, Fig. 4, we get a chart showing excess silica or negative silica, which if taken from the ore would make it self-fluxing. We have seen that in our slag, 1 part silica equals 2 parts lime, so if we read off our negative silica and multiply it by two, we get the amount of lime it requires. For example, on this chart an ore with 32 per cent silica has six points excess, or minus silica as shown by the lower silica line. Multiply by two gives 12 parts CaO to be added, which coincides with the amount of lime to be added, as shown by Fig. 3. Again, ore with 40 silica has fifteen points minus silica, which means 30 parts lime, which also checks out by Fig. 3.

We can in the same way plot mixtures of two or more ores in any fixed proportion. If the regular furnace charge be a mixture of two ores, the resultant mixture may be charted and a third ore figured as a flux. For example, if we have a mine A yielding silicious ore No. 1, and a mine B yielding an iron ore No. 2, we may decide on a standard mixture of, say, 1,000 pounds No. 1 to 2,000 pounds No. 2. This mixture can be charted and taken as a standard on which we may calculate the required amount of a limey ore No. 3.

Another interesting use of the graphic method is shown in Fig. 5. Suppose the standard furnace charge is 12,000 pounds of roast ore, and that this roast ore varies between 10 and 15 per cent sulphur; and suppose, further, that it is desirable to keep the sulphur contents constant at 13 per cent, and that we have a green ore containing 30 per cent sulphur with which to raise the amount of sulphur in the charge. We can lay out a chart showing just how much green ore must be added to roast ore of any sulphur content in order to raise the sulphur to any given percentage.

We start the calculation with some standard weights, for

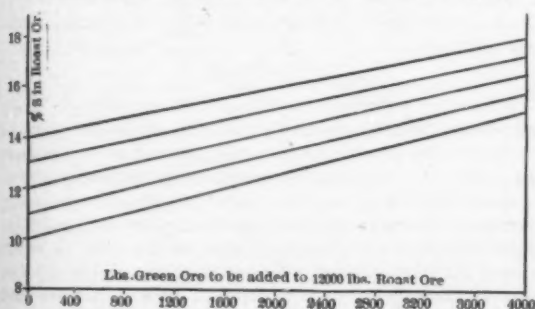


FIG. 5.—CHART FOR FURNACE CHARGE.

example: 12,000 pounds roast ore and 4,000 pounds green ore, as follows:

12,000 pounds roast ore at 10% S	= 1,200 pounds S
4,000 pounds green ore at 30% S	= 1,200 pounds S
16,000	2,400 = 15% S
12,000 pounds roast ore at 11% S	= 1,320 pounds S
4,000 pounds green ore at 30% S	= 1,200 pounds S
16,000	2,520 = 15.7% S
12,000 pounds roast ore at 12% S	= 1,440 pounds S
4,000 pounds green ore at 30% S	= 1,200 pounds S
16,000	2,640 = 16.5% S
12,000 pounds roast ore at 13% S	= 1,560 pounds S
4,000 pounds green ore at 30% S	= 1,200 pounds S
16,000	2,760 = 17.27% S
12,000 pounds roast ore at 14% S	= 1,680 pounds S
4,000 pounds green ore at 30% S	= 1,200 pounds S
16,000	2,880 = 17.9% S

Lay out on a vertical line at the left the different percentages of sulphur and along a horizontal line lay out the pounds of green ore—from 0 to 4,000 pounds. Take the 10 per cent sulphur which we find was raised to 15 per cent by the addition of 4,000 pounds of green ore, mark the point corresponding to 15 per cent S above 4,000 pounds of green ore; connect this with the point corresponding to 10 per cent sulphur with no green ore. This gives a diagonal line along which we can read the sulphur obtained by mixing 12,000

pounds roast ore at 10 per cent sulphur with any amount of green ore at 30 per cent sulphur. Connecting the other calculated points we have Fig. 5 completed and can read from it any desired mixture of green and roast ore.

The method of charting the work of a furnace from day to day, so as to obtain a graphic record of its consumption and output, and also to obtain a prophetic estimate of its probable tonnage for the

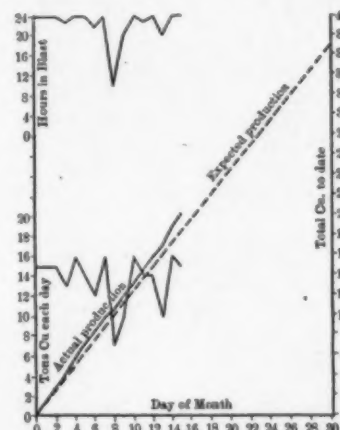


FIG. 6.—FURNACE CHART FOR A MONTH.

current month, is well known, but may bear repetition in this connection. Suppose a furnace smelting about 350 tons of ore and making about 12.3 tons of copper a day. If this furnace continues in blast every hour during the month we would expect 370 tons copper for this month. We lay off a horizontal line of thirty days, and on the thirtieth day erect a perpendicular upon which we mark a point corresponding to 370 tons, which we connect by a diagonal dotted line with the point on the base representing the beginning of the month. We graduate the right-hand vertical in a scale of tons from 0 to 370, and we graduate on the left a vertical column on a scale ten times as great for the daily output in tons. At the top of the left-hand column we lay off a scale of hours from 0 to 24, showing the number of hours the furnace was in blast each day. As we mark in each day's tonnage, we also mark in the total tonnage of copper produced up to date. This total tonnage to date, as long as the furnace produces 12.3 tons a day, will coincide with the dotted line of expected production but will deviate from this and fall either above or below it, according as the furnace produces more or less than the average amount of copper. Any delay or stoppage of the furnace is shown by the line representing hours in blast, and the cause of any delay may be marked by a circle enclosing a letter or figure referring to the usual causes of delay and explained in the legend. Such a chart from the 1st to the 15th of one month is shown in Fig. 6.

In the same way the output of the mines, the performance of day and night shift, the furnace records, tonnage of ores, flux, coke, analyses of the ore, amount or pressure of air may be tabulated from month to month. By plotting these on translucent paper, they can be laid one over the other for comparison, and in this way interesting relations and points of economy may be discovered.

For example, the monthly averages showing copper in the matte and copper in the slag should be parallel; if they are not, a comparison of the silica chart of the slag with the copper in the slag may show the cause of difference. Again, the matte grade and the life of the converter lining should be parallel; but if a better grade of matte in a certain month is not accompanied by an increased duration of lining, the explanation may be found in the analysis chart of the quartz

used which may show bad quartz for the month in question.

Often, also, the continuous relation of two lines may open a field for study. For example, the silica in the ore and the percentage of sulphur in the ore after roasting may be inverse one to the other; which on investigation proves that a certain amount of rock helps the roasting by cracking and swelling and so opening up the ore in the roast beds. This shows, also, the desirability of keeping a certain amount of rock in the ore.

But the output of the furnace per month may also prove to be exactly inverse to the silica in the ore; and this raises the question of volume of slag loss and the metallurgical balance sheet. Frequently by such comparison we see that what is apparently the cheapest method is in reality the most expensive. It might be thought cheaper to leave silicious rock in the ore, than to pick the rock out and afterwards add barren quartz for flux; but a comparison of the slag volumes will show that in many cases this apparent waste is an actual saving of expense.

There is no factor of furnace practice, except perhaps the personal equation, which cannot be studied by the graphic method. Even the personal equation may be referable to some disturbing condition. The number of furnace accidents may be shown to be greater under one foreman than another, or may reach a maximum at certain hours of the day, showing either individual carelessness or improper conditions under which the men are laboring. All knowledge comes by the intelligent comparison of data and the graphic method is the readiest means of making our data intelligible.

The Problems of Bleaching with Paper-Pulp Electrolytic Hypochlorite Solutions.

By W. POLLARD DIGBY.

Among the problems announced as suggested for general discussion at the forthcoming Philadelphia meeting of the American Electrochemical Society, is one worded as follows: "Is the Electrolytic Preparation of Bleaching Solution for Pulp a Success?" This announcement led the present writer to compile and collate some notes from his memoranda on the subject, some of which were recent and some made about eight years ago.

At the outset the question arises as to how success is to be defined. Ultimately the answer will have to be expressed in terms of a monetary system of exchange. But before this ultra-commercial stage of regarding the question is reached, a number of dependent matters call for consideration. Until the advent of the McDonald electrolyzer, described in the February issue of *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*, the problem was regarded as involving the choice of three alternatives, namely:

1. Calcium hypochlorite solutions prepared from chloride of lime.
2. Sodium hypochlorite solutions chemically prepared.
3. Sodium hypochlorite solutions electrolytically prepared.

The first of these undoubtedly holds the field. The second, which acts more rapidly, cannot hope to challenge its supremacy by reason of the waste of chlorine occurring during manufacture. In connection with the third, the writer is able to put forward in these columns some hitherto unpublished results.

Matters found to be of importance before any comparative data can be compiled are:

(a) Condition of pulp prior to bleaching. With wood pulps this is a constant factor for wood pulp from the same source; but where esparto grass is used its cleanliness is of great importance.

(b) Rapidity of action.

(c) Final color attained.

(d) Effect of bleaching medium on paper produced.

CLEANLINESS OF PULP.

Taking these in the order given, the writer has found considerable differences between paper mill and paper mill using, say, esparto from Tripoli. Some mills, for instance, where washing water is plentiful, perform the operation of washing the boiled grass and breaking it up in special beaters. Other mills wash the boiled grass, and break it up in the same potcher in which bleaching takes place. The requirements of the latter class of mill in available chlorine are from 30 per cent and upwards in excess of some other mills of the former class. Moreover, the amount of washing accorded in mills of the latter class is not always a constant quality. It would, therefore, be well if some defined degree of cleanliness could be adopted as a standard. The writer would suggest the use of a quartz glass prism of wedge shape, the wedge tapering in depth from 3 inches to zero, and having a length of, say, 20 inches. Upon its stand would be secured a scale divided into fifth of an inch, the wedge having at its back a slab of porcelain of dead white color. The wedge would be fitted with a glass stopper and filled with an aniline dye of brown color, the solution being so prepared that the white slab observed through the thickest end of the wedge agreed with the color of a control plate of brown earthenware of the exact shade of unwashed esparto. A rectangular test tube would be filled with water squeezed from the pulp, and moved along the white slab above the wedge until the slab, regarded through the tube, agreed in color with the slab as seen through the wedge. The point on the scale corresponding to the middle point on the test tube would represent the degree of cleanliness attained.

No comparative tests of different bleaching media can be regarded as absolute unless the cleanliness of the washed esparto pulps is identical. An allowance of a definite number of gallons per cwt. of esparto as it comes from the boilers is not sufficiently exact, and the method pursued by the present writer of test tube comparisons might have been improved along the lines suggested.

RAPIDITY OF ACTION.

The rapidity with which bleaching is effected depends, given pulp of a standard degree of cleanliness, upon the strength and quantity of the solution present, upon the nature of the base with which the available chlorine is allied, such as calcium, sodium or magnesium, upon any excess of such a basic hydrate, and upon the temperature during the process of bleaching. Chemically prepared sodium hypochlorite solutions of equal chlorine content are more rapid than calcium hypochlorite solutions, and electrolytically prepared sodium hypochlorite solutions still more rapid. This is probably due to their containing either no excess of alkali, or else such a small excess as is needed to ensure temporary stability. As regards temperature, the operation of bleaching is always expedited by heat, and temperatures within the potcher vary during hot bleaching from 90° to 110° F. Cold bleaching is resorted to frequently on Saturdays, when the bleached grass is likely to be left standing in the potchers over the week-end, when "hot" bleached solutions would have a tendency to revert to a yellowish shade.

FINAL COLOR ATTAINED.

When the bleaching operation is finished, that is to say, when a certain degree of whiteness has been attained and the available chlorine in the bleaching medium has been expended, the question arises as to how the whiteness secured should be described. From a number of reports I note such expressions as "good," "not quite up to mill standard," "white," "very good." All these expressions are somewhat relative, and the verbal expressions of the interested onlooker depend upon somewhat singular prejudices in regard to their outlook. Such visual basis is best met by taking a series of numbered pieces of glazed white china (differing in intensity) and recording those numbers which more nearly agree with the average product during any week, and the number of lots of each shade. If an agreed

standard of this nature is fixed in advancing, much subsequent argumentative debate, which hinges upon a biased eyesight on each side, may be obviated.

COLLECTION OF DATA DURING TESTS.

Having cleared up various preliminaries such as are outlined above, a series of tests may be commenced. Each test to be complete should describe (1) degree of cleanliness (if possible of the washed grass; if this is impossible the time allowed and amount of washing water in gallons should be stated); (2) the

ceeding readings taken at stated intervals during the test, together with thermometric readings;³ (6) final degree of whiteness attained.

If the above are noted, the quantity of available chlorine required per equivalent unit weight of dry grass will be established in an absolutely satisfactory and definite manner.

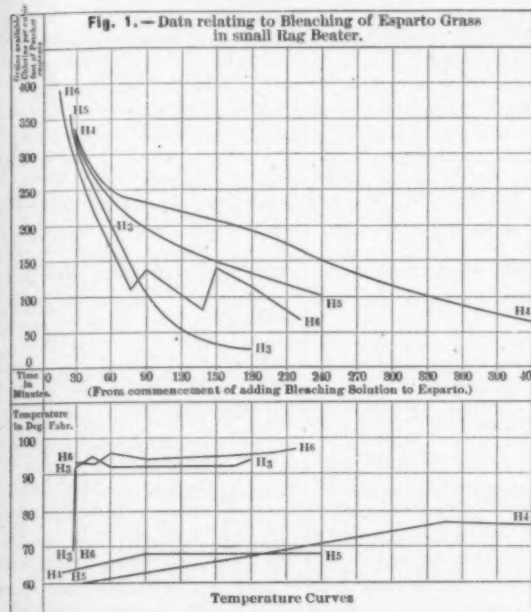
FIGURES OF ACTUAL SMALL SCALE TESTS IN A RAG BEATER.

Test No. H 3-5-4.—In this experiment 8 cwts. of wet esparto from the boilers (computed as equal to 4.16 cwts. dry grass) was bleached in a small rag beater. It was washed for 1 hour, but the quantity of washing water used was not recorded, the degree of cleanliness being checked by comparing the washing water at the end of this operation with a large potcher in the mill. The bleaching solution* consisted of 146.9 gallons, containing 458.8 grains of available chlorine per gallon. Fig. 1 records the fall in chlorine strength during the test. Immediately upon the completion of adding the bleaching liquid, steam was turned on and the temperature readings were recorded. The extreme rapidity with which bleaching was effected was very surprising. The color is reported as good.

Test No. H 4-5-4 was carried out immediately after that recorded above. It was intended that this should have been a "cold" bleach, but owing to a leakage in the valve controlling the steam inlet the temperature gradually rose until a temperature of 77° F. was reached. The mean average temperature being about 70° F., this test cannot be regarded as an absolutely cold bleach. With an initial chlorine strength identical with that used in the preceding test, a slower fall is recorded with a slower rate of change of color. After 3¼ hours the color was noted as "almost white," there being present 120 grains of available chlorine per cubic foot of potcher contents. A further 2¼ hours only caused this quantity to fall to 75 grains per cubic foot when a perfectly satisfactory whiteness was attained. It is apparent, therefore, that a surplus had been used. Allowing that this margin amounted to 2,000 grains the total quantity used would amount to 67,500 grains.

It was, therefore, decided to use less quantities of chlorine in succeeding tests, and to use a greater quantity of esparto, drawing off as much water as possible before adding the bleaching solution, so that the quantity of chlorine per cubic foot of potcher contents would be greater.

Test No. H 5-15-4.—The solution used consisted of 122.37 gallons of 540.64 grains av. cl. per gallon strength. The quantity of wet grass being 10 cwts. 2 quarters, computed as equal to 4.16 cwts. of wet grass. This test being carried out on a Saturday, it was decided to treat it as a cold bleach. The



quantity of esparto as obtained from the esparto boilers as wet esparto, together with the computed equivalent weight of dry grass;¹ (3) the quantity of bleaching agent supplied—as papermakers in England do not either use or think in metric equivalents, this may be expressed in gallons and the strength in grains of available chlorine per gallon of the liquor used; (4) the quantity of available chlorine present in the potcher upon the completion of adding the bleaching solution expressed in grains per cubic foot of potcher contents; (5) suc-

INDEX NUMBER OF TEST.	Actual Quantity Wet Esparto.	Computed Equivalent Dry Grass.	ELECTROLYTIC HYPOCHLORITE SOLUTION USED.		TIME REQUIRED.		TOTAL GRAINS AVAILABLE CHLORINE.		Grains Available Chlorine per Cubic Foot of Potcher Contents on Completion of Adding Electrolytic Solution.	Color of Product.
			Grains Av. Cl. per Gallon.	Total Grains Av. Cl.	From Commencement of Adding Solution.	From Completion of Adding Solution.	Per Cwt. Wet Esparto.	Per Cwt. Computed Equivalent Dry Grass.		
H3-5-4.....	Cwts. 8	Cwts. 4.16	458.8	147.	Hrs. 3	Min. 0	8,550	16,200	340	Good, but not up to mill standard.
H4-5-4.....	8	4.16	455.	133.	6	30	8,680	16,800	340	Good, but not up to mill standard.
H5-15-4.....	10.5	4.16	541.	122.4	3	55	6,300	15,900	360	Good, better than H3 or H4
H6-10-4.....	11.0	5.20	612.	104.	3	45	7,600	15,000	395	Very good.
			595.	24.						

¹ The term "computed equivalent weight of dry grass" does not mean the actual dead weight of grass present on evaporation of the liquid contents, but the arithmetical equivalent proportion of grass as actually fed into the boiler, of which a certain proportion of the weight is lost in the liquor drained off, and returned to the soda recovery plant. W. P. D.

² Thermometers, being fragile instruments, are not supplied for the use of mill laborers. The human hand is somewhat delicate, but in practice I have not found it a successful device, the margin of error being about 5 F. W. P. D.

extreme rapidity with which the whitening of the pulp was secured was strikingly marked. Initially there were about 350 grains of chlorine per cubic foot, and in 1½ hours, when the engine had stopped running, this had fallen to 135 grains. One and three-quarter hours afterwards there were still 100 grains of chlorine to the cubic foot, the color being fairly good. This continued to improve over Sunday, and on Monday the bleached pulp was worked up in the ordinary way.

Test No. H 6-19-4.—The amount of esparto treated amounted to 11 cwts. of wet grass, a higher initial chlorine strength being secured. The bleaching solution was added in one large and two small doses. The first dose consisted of 104 gallons of hypochlorite solution, containing 612.56 grains av. Cl. per gallon. The initial strength within the potcher when this quantity was added amounted to 394 grains av. cl. per cubic foot of potcher contents. In 75 minutes from the completion of adding the solution the amount of chlorine had fallen to 105 grains per cubic foot. The addition of 15 gallons of hypochlorite solution of 595 grains av. cl. per gallon strength, caused a marked rise in the amount of chlorine present. A further addition of 9 gallons was provided 60 minutes later. The test was completed in 3 hours and 45 minutes, giving a good white color 45 minutes before completion. Although on the cure the addition of the two subsequent small doses causes an appreciable rise in the quantity of chlorine immediately present in the potcher, these rises are due to the fact that the bulk of the bleaching had been effected, and that the extra quantities merely gave an enhanced whiteness to the paper.

SUMMARIZED RESULTS AND GENERAL CONCLUSIONS FROM SMALL SCALE TESTS.

From the figures given it is evident that the best results, as regards color and as regards quantity of chlorine required, were obtained when a solution of greater strength was used and when the fluidity of the material under treatment was reduced. This would tend to prove that if the washed grass had sufficient moisture extracted it would be possible to have an initially high chlorine strength within the potcher without an excessive amount dilution, or without the loss of electrical energy inseparable from the preparation of electrolytic solutions of high chlorine strength.

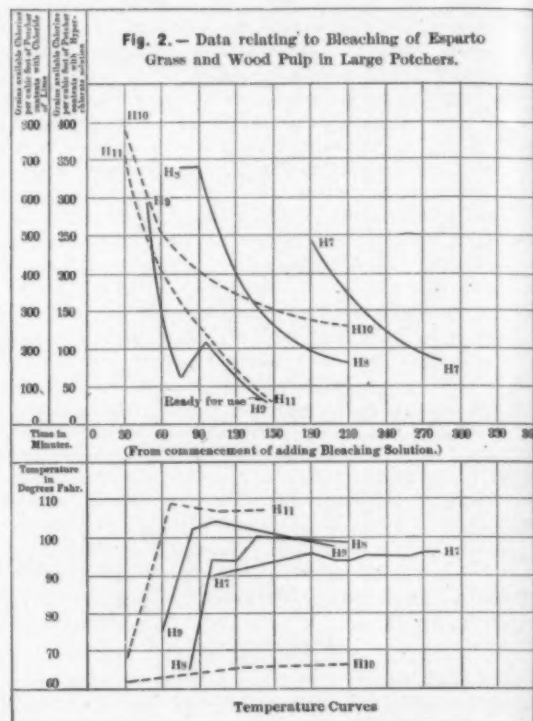
The small scale results being favorable, and the mill officials having pointed out that the use of the rag beater was in one respect unfavorable, namely, that the esparto was reduced to too fine a state of subdivision, and was therefore apt to be actually decomposed by the bleaching solution, it was decided to carry out five comparative tests in the large mill potchers. Three of these were made with electrolytic solutions and two with normal chloride of lime solutions.

FIGURES OF ACTUAL LARGE SCALE TESTS IN MILL POTCHERS.

Whereas, the previous tests all took place in a small rag beater upon esparto only, the large scale tests were upon a mixture of esparto and wood pulp. The quantity of the wood pulp cannot be stated in these columns. It was, however, constant in each case, and is therefore noted in the test records as the standard proportion of wood pulp.

Test No. H 7-28-4.—The amount of esparto treated amounted to 20 cwts. of wet grass, and to this was added the mill proportion of wood pulp. Owing to the small size of the pipe between the electrolyzer and the potcher, some 2 hours and 55 minutes were occupied in adding the hypochlorite solution. A

bleaching action was therefore taking place during this time, at the end of which there were 236 grains av. cl. present per cubic foot of potcher contents. The fall in the chlorine strength during the next hour and three-quarters is indicated in diagram No. 2. A good white color was obtained with the high quantity of 78 grains av. cl. per cubic foot of potcher contents unexpended. The total amount of hypochlorite solution used was 393 gallons, containing 600 grains av. cl. per gallon. This



quantity being in excess of the actual requirements, less quantities were subsequently used.

Test No. 8-4-5 was carried out upon an identical allowance of esparto and wood pulp. A less time was taken for adding the hypochlorite solution, only 70 minutes being so employed. The reason of curve H7 on the upper portion of diagram No. 2 remaining level for the first quarter of an hour, was due to the addition of a small quantity of solution (6 gallons during this time), after which the fall in chlorine strength took place at the usual rate. In 2¼ hours after the conclusion of adding the solution, the material in the potcher was bleached to a whiteness indistinguishable from that in the adjoining mill potchers using chloride of lime. An appreciable surplus of available

INDEX NUMBER OF TEST.	Actual Quantity Wet Esparto.	Computed Equivalent Dry Grass.	ELECTROLYTIC HYPOCHLORITE SOLUTION USED.		TIME REQUIRED.				TOTAL GRAINS AVAILABLE CHLORINE.		Grains Available Chlorine per Cubic Foot of Potcher Contents on Completion of Adding Electrolytic Solution.	Color of Product.
			Grains Av. Cl. per Gallon.	Total Grains Av. Cl.	From Commencement of Adding Solution.	From Completion of Adding Solution.	Per Cwt. Wet Esparto.	Per Cwt. Computed Equivalent Dry Grass.				
H7-28-4.....	Cwts. 20	Cwts. 8.65	600.	393.	Hrs. 4	Min. 40			11,780	27,200	240	Very good, better than H6. Very good, indistinguishable from mill samples. Very good, equal to H8.
H8-4-5.....	20	8.65	471.	421.	3	25			9,920	23,000	340	
H9-11-5.....	21	9.0	530.	381.	2	50			10,410	24,300	295	
			CHLORIDE OF LIME SOLUTION.									
H10-M.....	20	8.65	1400	220	3	0			15,400	35,600	790	Very good. Good, improved on standing over week-end.
H11-M.....	20	8.65	1400	220	3	45			15,400	35,600	800	

chlorine has again to be noticed. The total quantity of solution used amounted to 421 gallons of 471 gr. av. cl. strength.

Test No. H-9-11-5.—The total quantity of wet esparto, which came from Philadelphia instead of Tripoli, amounted to 21 cwt., and to this was added the standard amount of wood pulp. The bleaching solution was added in 50 minutes. During the next half hour an abrupt fall in the chlorine strength. This was so rapid that 31 gallons more solution was then added. In 2 hours from the start the material under treatment was of a very good color, and fit for running off for use. It was, however, kept in the potcher for another hour until a "chest" was ready, so that this batch could be made up separately. The total quantity of solution used amounted to 381.3 gallons of 530 grains av. cl. per gallon strength.

Test No. H-10-M.—For comparative purposes records were taken of the normal practice in the mill potchers using chloride of lime solution, and 20 cwt. of esparto plus the mill proportion of wood pulp. This was a hot test, the temperature rising to 107° F. Bleaching may be reckoned as beginning half an hour after commencing to add the solutions of chloride of lime, which amounted to 220 gallons, containing 1,400 gr. av. cl. per gallon.

Test No. H-11-M.—This was a "cold" test, taken with chloride of lime in the mill potchers, all quantities being identical with those in Test No. H-10-M.

SUMMARIZED RESULTS AND GENERAL CONCLUSIONS FROM LARGE SCALE TESTS.

Examining, in the first instance, Tests Nos. H-7, H-8 and H-9, in which electrolytic solutions of sodium hypochlorite were used, it is noteworthy that the best result was obtained when the chlorine contents of the potcher had the highest value immediately upon the completion of adding the bleaching fluid, and that this involved the smallest total consumption of chlorine per unit of dry esparto in the series. Further, this best result was attained with a solution of 471 grains per gallon strength, as against a solution of 600 grains per gallon strength in the test immediately preceding. It seems a reasonable inference, therefore, that an important factor is that of the amount of moisture present in the washed pulp before adding the bleaching liquid. This phenomenon is analogous to some which occur in the smelting of steel, the initial intensity of the chlorine present per unit volume of potcher contents, corresponding with the temperature attained in the metallurgical operation. Too low a temperature continued for a long time with a large thermal outlay would be practically without effect in the elimination of certain impurities, whereas a sufficiently high temperature for a shorter time, with a less thermal expenditure will secure the desired end. In bleaching too low a chlorine strength, even if maintained for a longer time, involving a larger consumption of available chlorine, fails to give the same grade of whiteness that can be secured where there is less initial moisture present with the fiber and the initial chlorine strength greater.

Quite apart, therefore, from questions of the cost of the solution and methods of production it is evident that the grass to be bleached should be as dry as possible, and this can be secured by squeezing the washed grass in a press.

Passing next to comparisons between the electrolytic solution of sodium hypochlorite and chloride of lime, the most striking feature is that the bleaching action of 1 grain of chlorine in the former is equal in intensity to 1.52 grains of chlorine in the latter; that appreciably lower temperatures may be used, about 10° F. less for hot solutions; and that the action is much more rapid for cold solutions. The difference in efficacy of the chlorine atom in the electrolytically prepared sodium hypochlorite molecule, as compared with the chlorine atom, the chemically prepared calcium hypochlorite molecule can only be explained on the hypothesis that the former is less stable in the presence of organic matter, and acting with a greater rapidity involves less loss through actual disintegration and decom-

position of the fiber than takes place with more stable substances.³

Having, however, determined for any paper mill the requirements in available chlorine for the mill potchers in regard to regular working conditions, the total requirements of the mill per day can be estimated. The question then passes from the hands of the chemist to those of the electrochemist and electrician.

THE COST OF PRODUCING HYPOCHLORITE SOLUTIONS ELECTROLYTICALLY.

This may be summed up roughly as so many kilowatt-hours of electrical energy, and so many pounds of sodium chloride per unit quantity of available chlorine produced. A rough approximation will not, however, suffice. What may be termed as the efficiency characteristic of the electrolyzer should be determined in regard to the units of electricity and weight of sodium chloride required per kilogramme of available chlorine produced for solutions of different strengths. As is well known, the initial high electrochemical efficiency falls as the strength of the solution rises, while the salt consumption falls continuously. The resultant cheapest cost of production depends upon the respective local costs for energy and salt. The efficiency and salt consumption characteristics of most electrolyzers are recorded in Engelhardt's "Hypochlorite und Elec-

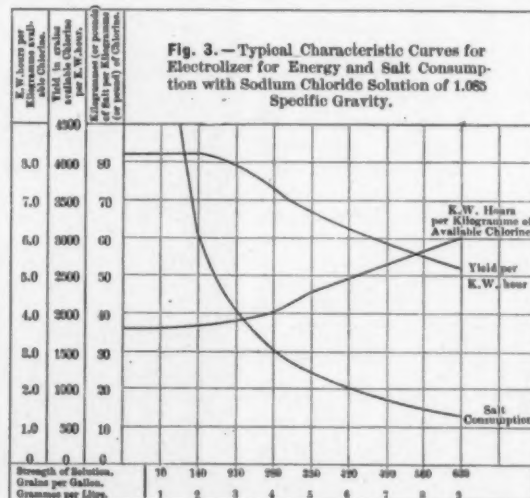


Fig. 3.—Typical Characteristic Curves for Electrolyzer for Energy and Salt Consumption with Sodium Chloride Solution of 1.085 Specific Gravity.

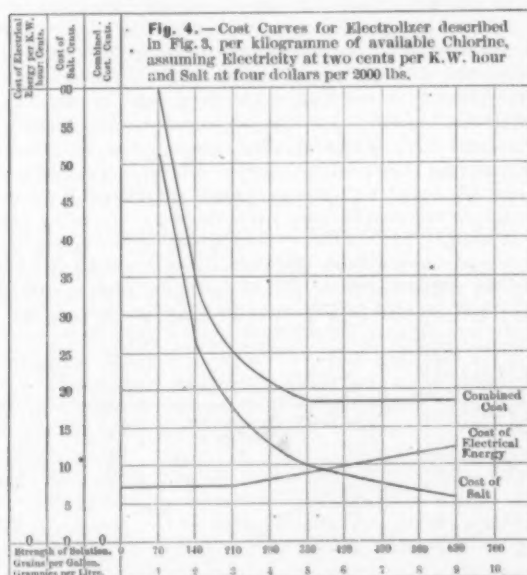
trische Bleiche," and in issues of the technical press appearing subsequently to the Engelhardt monograph. The writer presents in Fig. 3 the characteristic curves for the electrolyzer used by him in his experiments.

The application of its results to any paper mill depends, as has been said, upon local circumstances. In Fig. 4, hypothetical cost curves are reproduced for this electrolyzer upon a basis of electrical energy at 2 cents per kilowatt-hour, and for salt (i. e., sodium chloride) \$4.00 per ton of 2,000 pounds. As will be seen, minimum cheapness of production is secured under these circumstances for solutions of 350 grains available chlorine per gallon strength. If the salt cost were unchanged, electrical energy could be secured at a cheaper rate, the cheapest working point being at higher strengths, and an increase in the cost of electrical energy would shift the cheapest working point towards lower strength solution.

One important matter in the history of the production of electrolytic hypochlorites has been that the price of electrical energy has continued to fall, and with cheap power rates and the very good load factor which such processes offer, greater

³ It would be interesting to ascertain whether in the case of the calcium hypochlorite solution produced by the McDonald electrolyzer any difference exists either in regard to stability or in regard to requirements per unit quantity bleached, as compared with calcium hypochlorite solutions made in the ordinary way from chloride of lime. W. F. D.

attention has been of necessity diverted to reductions in the salt consumption. Primarily in electrolyzers of the Woolf, Hermite Crawford, Haas and Stahl, Vogelstein or Atkins types, this has been secured by methods of cooling, improved circulation and so forth, in order to secure liquids of a higher strength in available chlorine. A recent British patent (No. 21,949 of 1905) provides for the isolation of the respective anodic and cathodic products by means of diaphragms from the main body of the electrolyte, and their recombination in the electrode compartment, where one or other of these products is liberated in a nascent condition. Very great reductions in



the salt cost are effected thereby. In the McDonald electrolyzer, sodium hypochlorite is not produced in the main body of the electrolyte, but calcium hypochlorite is produced by the action of the chlorine gas upon a lime solution in chlorination towers.

For all electrolyzers in which the recombination of the anode and cathode products takes place in the main body of an electrolyte rich in salt, further salt savings than have yet been secured are necessary if such electrolyzers are to compete on the most favorable terms with chloride of lime. This can only be done if the electrolyte, after performing its operation of bleaching in the potchers, is squeezed out of the pulp, and after sedimentation and filtration returned to the electrolyzers. For this to be done properly the grass should be squeezed in a press after washing, and the proper weight of dry, clean grass put in a potcher and bleached with the hypochlorite solution. After bleaching it would have to be squeezed in another press. The salt loss by this process becomes merely a matter of the residual sodium chloride remaining in the interstices of the fibers of the pulp, and a recovery of from 70 to 80 per cent of salt becomes easily feasible.

A saving of this kind acts in two ways. In the first place, the salt consumption is reduced to three-tenths, or even two-fifths of that shown on the electrolyzer characteristic curve, and therefore obviously modifies the salt cost curve; and in the second place, modifies the combined cost curve in a manner permitting of working at lower chlorine strengths and with higher electrochemical efficiencies.

THE COMPARATIVE BALANCE SHEETS.

At this stage, when the cost curves of the electrolyzer are prepared from the efficiency characteristic curve and salt consumption of curve, upon local prices and conditions as to salt recovery, it becomes possible to discuss alternative balance

sheets and their composition. On the one hand, the mill consumption of chloride of lime per potcher, or per working week is known, and its cost free on rails at the mills obtained. To this cost must be added that of the labor required in handling the chloride of lime and mixing and pumping the solution, together with the cost of returning the empty casks and disposing of the insoluble lime sludge. On the other hand, the consumption of electrolytic hypochlorite having been ascertained, and its cost for electrical energy and salt known, there must be added such items as interest on capital outlay, wear and tear on generating plant (if one has been installed), wear and tear of electrolyzers, together with the energy consumption required by such auxiliary machinery as may be used for mixing the saline solution, cooling the electrolyte, filtering the returned saline liquor from the potchers, as well as the cost of operating the presses which squeeze the moisture from the pulp, together with the labor required.

GENERAL CONCLUSIONS.

The investigations which the present writer has conducted, both in the laboratory and in large scale experiments, lead him to believe very strongly in the possibilities of electrolytic hypochlorite processes for paper and textile bleaching. It is necessary for effective bleaching that the initial strength of available chlorine within the potcher should not be less than 350 grains per cubic foot, and it does not at present seem necessary that this value should exceed 500 grains. The intensity of the whiteness attained depends largely upon this initial strength and upon the material not being too fluid in its nature. With pulp free from too much moisture, solutions containing from 3.0 to 4.0 grammes of available chlorine per liter can be produced at a high electrochemical efficiency. The electrolytic solutions act with greater rapidity, and the degree of heating need not be pushed so far. Above all, there remains the fact that one atom of chlorine in the sodium hypochlorite molecule in the electrolytic solution is more potent than one atom of chlorine in the calcium hypochlorite molecule of a chloride of lime solution. In the case to which the present figures refer this margin was found to be 1.5 to 1.0. In the case of another mill, the writer estimated it at 1.65 to 1.0, and a consulting chemist, to whom the mill owners referred the question, reported from his laboratory tests that the figure was nearly 2.0 to 1.0.

Finally, the writer would strongly deprecate the application of these figures which are true of one mill, or of any other figures equally true of another mill, to any and every case which may transpire. Each case requires detailed local examination before any declarations are made.

Pierre Eugene Marcelin Berthelot¹.

BY CHARLES A. DOREMUS.

France, for the third time in the space of a few months, mourns the loss of an illustrious savant. Again the nations of the world extend their sympathy. The heroes acclaimed were not martial victors over mankind, whose honors were bought in the price of blood, but men of lofty ideals who conquered nature, brought truth to light, instituted new industries and improved old ones, thus bettering man's physical, and through enlightened thought elevating his moral condition. Truly "science guides humanity."

Curie, Moissan, Berthelot! How varied the achievements of each, though each chose chemistry as a field of labor. How differently each worked out his task and how successfully.

Berthelot was born in Paris, Oct. 25, 1827. His father was a physician, and the young man inherited not only a taste for

¹ Read at the meeting of the New York Section of the American Chemical Society, April 5, 1907.

a scientific career, but was schooled most effectually for it. His education at the Lycée Henri IV. developed the taste for historical research which won him his first prize and which later directed his attention to the early history of alchemy, the foundations of our science. His great erudition, his mastery of the Greek language and his love of exactitude in securing fundamental facts have given us nine volumes covering the several topics of these researches. They appeared from 1885-1893 and represent his maturer years when the activities and acquisitions of middle life were subjected to the criticism of a calm judgment. This phase of Berthelot's character is also seen in his frequent minor articles dealing with questions of education, morals and philosophy. He possessed a marvelous memory. He lived in a period when the sciences were rapidly developed. He obtained an extraordinary grasp of their relationship. He lived in an environment which was stimulating. He quickly understood what was fundamental in each, and so at eighty he was one of that type of men, now growing rare because of the intense specialization of our day, known as "the encyclopedists."

His first scientific memoir was presented to the Académie des Sciences May 27, 1850. It described the liquefaction of gases by the pressure secured by the dilatation of mercury. He found that pressure alone would not reduce gases to the liquid state. From that date there was no cessation in his labors; he attended, as its perpetual secretary, a meeting of the Académie within an hour of his death, March 18, 1907.

He became assistant to Balard at the Collège de France and obtained his doctorate in 1854, with a sensational thesis on the synthesis of natural fats from glycerin and the fatty acids. A continuation of these researches, especially on the polyatomic alcohols led, in 1863, to the founding of the chair organic chemistry at the Collège de France, that he might have the conditions for carrying out his personal ideas. He thus entered on a field of work which made him famous. Analysis had until this period been the chemist's

aim. *Synthesis* now claimed his attention, and before the end of the nineteenth century wonders were indeed wrought; revolutionizing both philosophy and the arts.

By causing an electric arc to play between carbon electrodes in an atmosphere of hydrogen Berthelot secured the direct union of carbon and hydrogen with the production of acetylene. He then converted this by the action of heat into benzene, and from these passed to other syntheses. He also experimented with the silent discharge turning oxygen to ozone. With the induction current he combined acetylene and nitrogen to hydrocyanic acid. He obtained formic acid starting from carbon monoxide. By the use of sealed tubes in which chemicals were subjected to high temperature and pressure through considerable time he influenced them to combine, and also gave us a new general method in chemical manipulation. Six important works, in all nine volumes, attest his genius as applied to this department of his labors. His soul was in his work. When one contemplates how his experiments steadily progressed, effecting the grouping of the elements to form hydrocarbons, alcohols, acids, ethers, sugars, fats, thus simulating natural processes and building up compounds which up to his day were conceived as being solely the result of vital force, we little

wonder that he became permeated with the idea that ultimately man would manufacture his own sustenance. In his address to the second International Congress of Applied Chemistry he says: "No one can deny that the day is perhaps near when the progress of chemistry will realize the manufacture of foods; in that day the cultivation of wheat and the raising of cattle will be exposed to the same destiny which has overtaken the culture of madder in our day." What perplexing situations will then arise with reference to the pure food law!

Berthelot took an active part in the great movement of the middle of the nineteenth century when the correlation of the sciences was discussed and the conservation of energy was established as the basic principle in physics. It was, therefore, natural that he should attempt to measure the energy developed by chemical reactions in definite terms. He labored indefatigably for thirty-five years in founding thermochemistry. The facts and principles are collected in two volumes published in 1897. They had been preceded by a work entitled "Essai de Mécanique Chimique," also in two volumes, 1879, followed by a volume, "Traité pratique de Calorimétrie Chimique," 1893. His two volumes "Sur la force des matières explosives," 1883, was intimately connected with these other

laborious researches in thermochemistry, and led to the discovery by others of smokeless powder. His work on the detonation of endothermic substances, such as cyanogen and acetylene, was followed by a research on explosive waves by which he elucidated many seeming contradictory facts. During the stirring times of 1870 Berthelot was made president of the scientific committee on defense; he afterwards became consulting member of the committee on powder and saltpetre, and president of the commission on explosives. In connection with these duties he devised many original methods of research.

His thesis that chemical phenomena are identical in animate and inanimate nature is thus expounded in 1855: "We may, I say, claim to form anew all the substances which have been developed since the origin of

things, to form them under like conditions in virtue of the same laws, by the same forces which nature brings into play in their formation."

And as a necessary sequel of his life's work we find him attacking the serious problems of the theory of agriculture and of biological chemistry. His "Chimie Agricole," in four volumes, and his "Chimie Animale," in two volumes, were both published in 1899.

The beautiful experiment farm at Meudon was the scene of his labors. One climbs the "tour Berthelot" of over eighty feet in height and about one is the charming scenery of this suburb overlooking Paris. Here the master undertook his experiments on the influence of electricity on the growth of plants—generating this force or deriving it from the atmosphere. Here it was that the fixation of nitrogen was studied, a problem that has engaged the sturdiest minds, and here it was that he found that microbic life was the means of transferring atmospheric nitrogen to the living plant cell. The import of this phenomenon he tersely stated in saying: "The soil is something alive!" To us the products of whose broad acres furnish enough for ourselves and to spare this discovery is of incalculable value. And yet this man, who instructs us in eco-



PROF. AND MME. BERTHELOT.

nomie farming, does not hesitate to indicate how we may manufacture our own food and thus make ourselves independent of climatic influences.

The experimental investigation on plant life led to that of the animal organism. The principles of the production of heat in living beings was a topic quite germane to the investigations on thermochemistry.

While Berthelot found his greatest pleasure in experimentation in science he was fully alert to the intimate relations his investigations bore to the advancement of the liberal arts. He made his position in this regard quite clear to the audience he addressed at the second International Congress of Applied Chemistry:

"In chemistry, as in all studies useful to man, theory and practice are related to each other by indissoluble bonds.

"Senseless the theorist who, shutting himself up in the solitude of his egotistical personal views, affects to disdain the incessant applications of science to civilization, for the wealth and happiness of mankind!

"Senseless, no less senseless, the practical man who, satisfied with the knowledge acquired by his ancestors, out of admiration for their conservatism and tradition, opposes all progress, refuses to enlarge or change the processes used in his industry, that it may remain each day in complete accord with the newest and most advanced theory!

"No science probably, more than chemistry, shows the necessity of this constantly renewed harmonious relation between practice and theory."

To-day the traffic of this great city, the incessant tide of travel, the lighting of its streets and homes, is effected by the aid of electricity generated by the burning of coal, and the specifications, under which the coal is bought, require that its calorific value shall be determined by the bomb calorimeter, invention of Berthelot, devised for theoretical purposes.

While a great theorist, he invariably had recourse to the experimental method for establishing his premises on a sure foundation. His temperament was that of an idealist, of a literateur, yet he foreswore in part his allegiance to science to serve his country.

France had honored him by the bestowal of many favors in recognition of his labors. Member of the Institut, 1900, grand officer of the Legion of Honor, 1886, perpetual secretary of the Academy of Sciences, 1889, succeeding Pasteur; but she also made him senator for life, 1881, gave him the portfolio of the Minister of Public Instruction and the Fine Arts, 1886, and made him Minister of Foreign Affairs, 1895, and he served his country with ardor.

The fiftieth anniversary of his first scientific publication was celebrated at the Sorbonne on Nov. 24, 1901. Official delegates of foreign scientific societies voiced their congratulations. The French Academy, in a stirring discourse delivered by Moissan, "tendered him its homage and thanked him for having given it a little more of truth." All departments of the government were represented at this unique festival. A beautiful medal, by Chaplain, bore on its face the likeness of Berthelot and the inscription, "La Synthèse Chimique. La Science Guide l'Humain." On the reverse side the savant appears seated before his laboratory table, on which is placed new classical apparatus, while above are two figures typifying the inscription "Pour la Patrie et la Vérité," and the president of the republic, M. Loubet, as he handed him this gift, kissed the dear old man in token of the love and gratitude of the nation and in behalf of his admirers of all other nations.

Berthelot was particularly happy in his surroundings. He was constantly in his laboratories in Paris, Meudon and elsewhere; it was here that his *positive* science claimed him. In late years he resided in the Institut, a palace formerly occupied by Cardinal Mazarin. It was here, surrounded by his family and friends, that he enjoyed his *ideal* science.

He married early in life a beautiful and charming woman by whom he had five children, the four sons surviving. The forty-

five years of married life came to a dramatic end. Both husband and wife suffered from heart trouble. Berthelot, anxious about his partner's failing health, was ever watchful. He left her to be present at the semi-monthly meeting of the academy, but returned shortly—only in time, however, to be with the beloved one in her last moments. Shattered by the blow he was led to a couch in his work room. Alas! The strain had been too great and his own heart, weakened by age and the present anguish, ceased beating.

On March 25 this noble man and woman were given public obsequies. The great Pantheon was filled with the representatives of all branches of the government from President Fallières down. The edifice was crowded with distinguished men and women. As the two bodies rested on catafalques M. Briand gave an eloquent discourse. Afterwards the body of Berthelot was placed on another catafalque before the church and the army passed in review, saluting the great dead. In the afternoon the public did him homage, and towards evening he and his dear wife were placed in the crypt, not far from the remains of Victor Hugo.

In his peroration to the second congress Berthelot summed up his views of life; he fulfilled them in his own: "Our duty is clearly outlined. Let us be doing, that is let us work! Work without cessation; let us try to be useful. Diligence and the love of mankind! This is the true aim of both home and public life."

The Moore Vacuum-Tube Light and the Luminosity of Gases

By C. J. THATCHER, PH. D.

The past few years which have witnessed a rapid improvement in the arc and incandescent forms of electric lighting have also seen the commercial application of a new and distinct type of lamp; that is, the enclosed vapor or vacuum-tube lamp. Three forms of this type have become generally known. The first of these is the Cooper Hewitt, which is now widely used and well understood; a second is the Bastian, which like the former is a mercury vapor arc lamp, but of smaller dimensions and provided with an incandescent carbon filament lamp to provide red light rays; the third is the Moore vacuum tube. Since the latter has recently given evidences of commercial practicability and is perhaps not thoroughly understood, a brief description of it and consideration of its salient features will be given here. It will be shown, also, that certain phenomena observed in this tube may have an important bearing on the question of the nature of gaseous luminosity in general and one which has not been previously recognized.

Mr. D. McFarlane Moore began to experiment about twelve years ago for the purpose of producing a "cold light," that is, one in which all the electrical energy would be converted into light without any loss in the form of heat. At that time the maximum lighting efficiency was obtained with the arc lamp, consuming 1 watt per candle power. Dividing this value into the theoretical figure for the conversion of electrical energy into light as given by Dr. E. F. Roemer—one spherical candle power per 0.115 watt—we see that only about 10 per cent of the electrical energy is converted into light in this form of lamp, the other 90 per cent is dissipated and lost in the undesirable form of heat.

Since then, however, illuminating art has been much improved, so that we now have a maximum lighting efficiency of 50 per cent in the flaming arc lamp, which consumes only 0.23 watt per candle power. The most efficient form of the Moore light, that is, the one in which rose-red light rays predominate, has a maximum efficiency of about $\frac{3}{4}$ watt per candle, and is but little better than that of the older form of

¹ Transactions of the American Electrochemical Society, vol. viii, page 248.

are light. The heat loss is, therefore, about 85 per cent in the most efficient form of the vacuum tube lamp.

DESCRIPTION OF THE MOORE VACUUM TUBE.

Although the "cold light" problem therefore is still unsolved, Mr. Moore's experiments have finally produced a novel and fairly efficient form of illumination, and one which possesses distinct advantages for some purposes. As it is now being installed it consists of a continuous stretch of $1\frac{3}{4}$ -inch glass tubing of any desired length; this is supported near the ceiling by suitable brackets and encircles the room or space to be illuminated.

This continuous length of tubing is made *in situ* by joining together 6 or 8-foot lengths, and at the corners previously shaped angle pieces. The joints of the tube are made in the manner long used for joining large bore glass tubing in making chemical and physical apparatus; the blow-pipe employed is a slightly modified and movable form of that, having two impinging flames, which has long been used for that purpose.

The ends of the one-piece tube thus formed come together in a box about 2 feet square, which is suitably and inconspicuously placed. These ends are constructed of slightly larger bore tubing for a foot or so, and are, of course, rounded and sealed at their extremity. They each contain a electrode of carbon electrically connected with the exterior by platinum wires sealed in the end of the tube.

These wires in turn are in electrical connection with a transformer, which raises the voltage of the alternating-current supply to 10,000, at which pressure the current is delivered to the tube.

Another essential part of the system, also placed within the box, is an ingenious vacuum regulator. This consists of a cone-shaped carbon pencil, sealed point upwards in the end of a narrow glass tube, sealed in turn to the main tube. Surrounding this carbon pencil and sealed to the narrow inlet tube which carries it, is a glass tube of larger diameter. The annular space thus formed contains enough mercury to cover the entire pencil, and in this mercury a metal tube displacer floats, attached at its upper end to the core of a solenoid placed above the regulator and in the lighting circuit.

The tube thus completed is evacuated by a Geryk or other mechanical vacuum pump to a pressure stated to be about $1/40,000$ of an atmosphere. With the passage of the high-tension alternating current the tube immediately becomes luminous throughout, the light being a soft one with rose-red rays predominating.

A suitable manometer was first attached to similar tubes several years ago, which, however, were not then provided with a pressure regulator. It was then found that the rarefaction of the gases in the tube slowly increased, and this was accompanied by perplexing resistance changes, which finally culminated in the failure of the light.

In order to secure a permanent light it has been found necessary, therefore, to admit very small portions of air to the tube at intervals. This is accomplished by the vacuum regulator which was devised subsequent to the measurement of pressure variations. When the air pressure decreases the internal tube resistance also decreases; more current flows through the solenoid of the regulator, and the core thereupon rises a trifle, withdrawing the displacer so that the mercury falls enough to uncover the tip of the carbon pencil.

This is thereby exposed to atmospheric pressure, and a small bubble of air, to which the pencil is not impervious as it is to mercury, filters through the carbon; the normal vacuum and resistance are thereby restored and the mercury rises again, shutting off the air supply. Practically normal tube conditions can be thus maintained indefinitely, and tubes thus automatically controlled have had a life of 1,000 hours or more.

The circuit is conducted through the Moore tube from terminal to terminal solely by the highly rarified gases, and these alone are the source of the light it emits. The Moore light,

therefore, is only a lengthened Geissler tube; its light may at times exhibit striated effects similar to those of this tube, but ordinarily these are not noticed by the eye.

Mr. Moore has, therefore, made a practical application of a mode of producing luminous effects which has long been known; which, indeed, was one of the earliest forms of light produced by electricity. Now that this form of light bids fair to have a commercial value it is to be expected that the nature of it may be the subject of more frequent, or at least more successful, scientific investigation, for strange to say little is definitely known concerning the source of the luminosity of conducting rarified gases.

THEORIES OF GASEOUS LUMINOSITY.

It will be interesting to note here the theories regarding luminiferous gases recently advanced by different investigators in this field. The older theory is that the gaseous molecules send out vibrations of a visible wave length as a result of purely physical collision or heat effects consequent on electrical conductivity. Regarding this theory, Prof. H. A. Armstrong, in 1902, in an article on "The Conditions Determinative of Change and of Electrical Conductivity in Gases, and on the Phenomena Luminosity," stated: "An argument which I think will sooner or later be regarded as of weight in favor of the view that the phenomena are electrolytic in their origin is afforded by the luminous manifestations in vacuum tubes. These can scarcely be mere collision effects or mere heat effects. It has long seemed to me that luminosity and line spectra are the expression—the visible signs—of the changes attending the formation of molecules from their atoms, or, speaking generally, that they are the consequence of chemical changes, a chemical change being one which involves an alteration of molecular composition, or it may be of molecular configuration, as it is conceivable that even changes involving isodynamic (tautomeric) molecules—changes in molecular structure unattended with change in molecular size—may give rise to such manifestations." Prof. Armstrong believes gaseous luminosity to be the result of chemical activity therefore.

Prof. J. Stark, of Göttingen, on the other hand, who has made extended researches in this field, believes it to be a purely physical phenomenon, according to the following translated statement: "The line spectrum of a gas has its source of energy in the kinetic energy of the particles of the ionized gas, its conduction in the positive ion atoms. * * * The band spectrum of a gas, on the other hand, probably has its source of energy in the potential energy which the positive and negative ions possess in respect to one another, and which on recombination can be changed, at least partly, into light energy."

But a third authority, Dr. C. P. Steinmetz, in an address before the American Institute of Electrical Engineers, on "The Transformation of Electric Power Into Light," in November, 1906, stated regarding Geissler and vacuum tubes: "The mechanism of this light production does not seem to be known, but the light seems to be somewhat of the character of a by-product."

WHAT MAY BE LEARNED FROM THE MOORE LIGHT CONCERNING LUMINOUS GASES.

It has not been recognized, so far as I am aware, that the observed constant decrease of gaseous pressure in the Moore tube may have an important bearing on the question as to the cause of the luminosity of the gases contained in the tube, and, indeed, of conducting gases in general. As has been already stated, it has been found necessary to feed the tube with air at frequent intervals. As a matter of fact very small portions

² Royal Society of London, Proceedings 1902, p. 108.

³ Physikalische Zeitschrift, 1906, p. 764.

⁴ Proceedings of Amer. Inst. Elec. Eng., Nov. 1906, page 767.

are added every minute or so during use. This, of course, can mean nothing else but that chemical reactions are occurring which result in an increase of molecular size, and therefore in a decrease in the number of molecules and of the volume of gas.

The substances in the tubes in considerable amounts which might cause such reactions are, of course, nitrogen and oxygen and the carbon of the electrodes. Of the known reactions into which these might enter, those involving cyanogen formation, either as an intermediate or as an end product, cannot cause reduced pressure. The formation of ozone from oxygen, on the other hand, would cause reduction in the number of molecules and a higher vacuum. Ozone, however, is rapidly decomposed into oxygen again at temperatures below that of the interior of the Moore tube when in use, so that ozone cannot be a stable end product, nor can its formation cause the reduced pressure.

This leaves only the uniting of nitrogen and oxygen to form one of the oxides of nitrogen as the probable cause of the phenomena. The formation of NO and of N_2O_3 as end products would not result in reduced pressure, while that of N_2O , NO_2 or of N_2O_4 would. Of these, NO_2 is the more probable end product, but this investigation must, of course, verify.

It is true that solid brownish-colored deposits are frequently formed in the body of the tube; but is highly improbable that they are the products of reactions causing the observed gaseous contraction, for they are by no means invariably formed after prolonged use, and may, indeed, appear either very soon after installation or not at all. They are in all probability due, therefore, to impurities or variations in the composition of the terminals or of the glass tubing.

It seems certain, therefore, that the formation of one or more of these oxides of nitrogen is the cause of the reduced pressure of the tube. Apparently, Mr. Moore has come to the same conclusion—it may be by investigation—for he has often spoken of his light as one which "burns air."

Now, numerous investigations on the fixation of atmospheric nitrogen have demonstrated that the formation of oxides of nitrogen are purely thermal effects. (See Prof. Guye, this journal, 1906, p. 136, and Foerster and Nernst, *Ibid*, p. 256.) This being the case oxides of nitrogen will be formed in the tubes wherever there is sufficient heat development, and that is throughout the tube, since 85 per cent of the electrical energy put into the tube is converted into heat. Throughout its entire extent it becomes uniformly hot.

The heat development of the terminals is not appreciably greater than in the body of a tube, though bolometric measurements may show some slight difference. But this is certainly not considerable; so that in view of the far greater reacting volume of gas in the body of the tube it appears that the formation of oxides of nitrogen takes place in the Moore tube throughout its entire extent where it is accompanied by the emission of light.

The chemical actions here involved may not necessarily be direct ones. Ozone or other unstable but active molecular complexes may be concerned in it, and the reactions may be reversible and have low reaction velocities, but the sum total result of electrical activity in the Moore tube is, apparently, the formation of a stable oxide of nitrogen accompanied by and intimately connected with the continuous emission of light rays.

This conclusion is important in that its verification will furnish a practical substantiation of the theory of the chemical nature of gaseous luminosity advanced by Prof. Armstrong and already cited in this article. And verification is not difficult, it includes two steps: First, identification of the substances formed in a tube and an investigation whether the concerned reaction velocities are greatly influenced by heat. Second, an observation of the effect of temperature variations in the conducting gases on the candle power of the light which they emit, these temperature variations to be controlled by

means external to the tube. If, for example, NO_2 is formed, and if the reactions producing it give rise to luminous effects, then cooling the tube to a very low temperature during its activity, *e. g.*, by liquid air, would lower the candle power, for the velocity of NO_2 formation is greatly retarded by reduced temperature (see Guye, Foerster and Nernst, *loc. cit.*).

The probable result of such an experiment is indicated by one cited by Prof. Armstrong. He states that Prof. Dewar, in an experiment before the Royal Institute, cooled a phosphorescent Crookes tube with liquid air and that its discharge at once ceased. Prof. Armstrong attributes this result to catalytic rather than thermal effects. If cooling to a low temperature produces similar effects in the Moore tube, as from the close analogy it may be expected it will, the dependence of the luminosity of a mixture of conducting gases on the velocities of gaseous chemical reactions will have been proven. This could not before have been proven, because only a vacuum tube as extensive as the Moore tube could give reaction products in amounts which could be identified.

Experiments such as suggested, if they verify the theory outlined above, will also teach several things about the Moore tube and vacuum-tube lamps in general. The first of these is, that in order to secure high-efficiency gaseous light-emitting reactions must be employed which are exothermic or whose reaction velocities possess a relatively low temperature coefficient. That is, it must not be necessary to change much of the electrical energy into heat in order to maintain the high temperature for a rapid reaction velocity, as is necessary in the formation of oxides of nitrogen. Or if suitable reactions of this nature cannot be found the necessary acceleration of reaction velocity must be secured by the use of catalytic agents.

The second conclusion to be drawn from the above theory—an inference, indeed, which is palpably evident from what is already known concerning the Moore tube—is that there is a limit to its life. Air is constantly admitted to the tube, enters into chemical reaction therein, and the reaction products remain in the tube. If gaseous they will gradually accumulate therefore, and ultimately extinguish the light by displacement of the active gases; even if they were solid they would coat the interior of the tube and make the light very inefficient. It is to be expected that the Moore tube will occasionally need repair, therefore, though the contrary seems to have been stated.

ADVANTAGES AND DEFECTS OF THE MOORE VACUUM TUBE LIGHT.

Mr. Moore claims for his light a high efficiency, good actinic value, low intrinsic brilliancy, safety, perfect illumination without shadows and a very long life, indeed, that it will last for years without repair. The efficiency of the tube has recently been the subject of careful photometric tests by the New York Electrical Testing Laboratories, which showed that the average lucas per unit energy were 20.0 for the Moore, 11.2 for the Nernst, and 3.6 for the carbon-filament incandescent lights, which is equal to .65, 1.1 and 3.5 watts per candle power, respectively. This efficiency is for the rose-red light only. In tubes emitting white light the efficiency is stated to be only about half the above, that is, 1.5 watts per candle power. This is not as high an efficiency, therefore, as is obtained with the metallic-filament incandescent lamps now being introduced.

Mr. Moore at the present time rarely installs tubes giving a white light, probably because of their lower efficiency and unreliability; it is necessary to introduce other substances, and the tube conditions favorable to a long life or even satisfactory service are not yet thoroughly understood. The white light only is suitable for general use, of course, but the rose-red tint is satisfactory for exterior lighting and in rooms where proper color values are not essential.

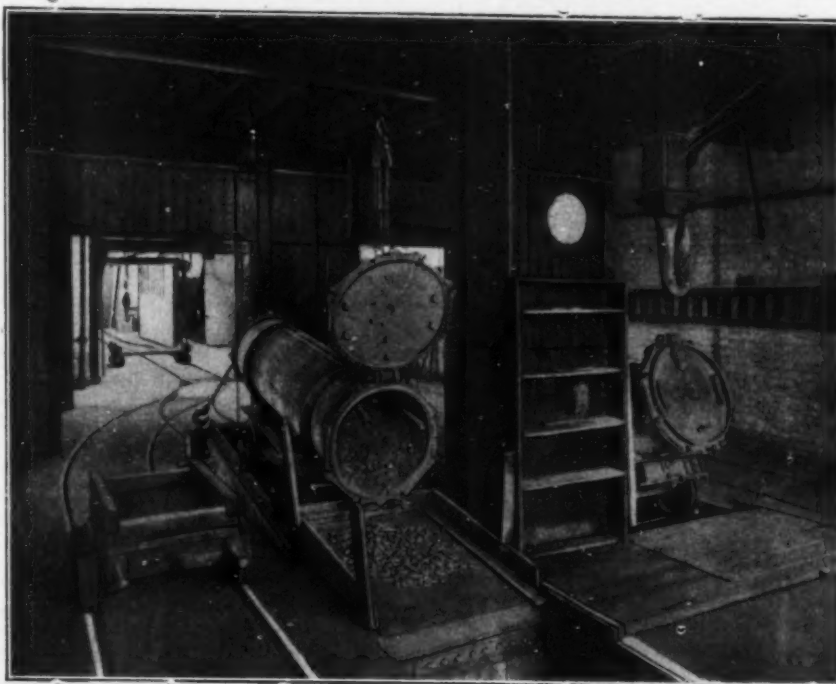
The principal advantage of the vacuum tube is its low intrinsic brilliancy. Its light is but 12-candle power per linear

foot. It has no extremely bright portion common to nearly every other form of artificial light, which strain the eyes whenever they are in the angle of vision. This evil is now being lessened somewhat by the general use of frosted globes or shades, but even these do not bring these other forms down to as low an intrinsic brilliancy as that of the Moore tube.

The perfect illumination without shadows, that is, from every angle of the room, which has been claimed as an exclusive advantage of the Moore tube is equally well secured by the use of small units like low candle-power incandescent lamps suitably placed entirely around the sides of the room or on the ceiling at intervals of a foot or so. This method of

necessary to produce an alloy, the extensive work of Roberts-Austin, Spring and many other investigators shows very conclusively that the inter-solution of metals may take place at any temperature. Alloying at a low temperature has the advantage of preventing the formation of eutectics, when particles of one metal are merely suspended in the other, and the internal stresses due to the uneven contraction of the mixture makes it valueless for most commercial purposes; the failure of bearing metals can, in most cases, be attributed to a complete or partial eutectic structure, due to careless manufacture.

If two metals in contact are both readily decomposed by the same agency, and if when in contact they are exposed to that



SHERARDIZING PLANT AT WILLESDEN, ENGLAND.

(Drums shown in loading and unloading positions. Note overhead zinc dust bin and screen upon which small articles are dumped, the dust falling through.)

installation produces the freedom from shadows quite as well as does the Moore tube.

The tubes are difficult to repair, and leave the room entirely without lighting facilities for a long time in case of a failure or accident to any part of the system. This fact, the high initial cost, and its want of what may be termed flexibility, that is, the impossibility of turning off or on at will a part of the tube illuminating any desired portion of the room, will militate against the rapid adoption of the Moore lamp for general illuminating purposes.

Theory and Practice of Sherardizing.

BY ALFRED SANG.

If a disc of sheet rubber and a disc of sheet copper are placed in contact for a few weeks, the surface of the copper is found to have a heavy blue coating of sulphate of copper. The sulphur used in the vulcanizing of the rubber has, by simple contact, combined chemically with the copper. If for the rubber disc we substitute a disc of zinc, and insure perfect contact by pressure, we shall, after a few months, find the surface of the copper covered with a film of yellow brass.

An alloy is the combination of two or more metals, and while a great many persons still cling to the idea that fusion is

agency, the metal which stands lower down the scale of electrical conductivity will be decomposed in preference to the other. This is the reason for the efficiency of a zinc coating for iron and steel articles and is distinct from the protection afforded by the covering as such. Moist air containing carbonic dioxide will oxidize both zinc and iron, but when they are in contact the zinc will bear the brunt of the attack. The closer the contact between the two metals the more effectual the galvanic protection will be, and the greater the exposed surface of the zinc is the more efficiently will it perform its self-sacrificing duty.

A heavy coating of zinc is not necessarily better as a protection against rust than a light coating. The thickness of the coating has little or nothing to do with the actual (galvanic) efficiency; it has, however, some bearing on the mechanical efficiency, and must receive consideration (apart from the Preece test) when, either the zinc oxide is constantly removed, exposing a clean surface to the action of oxidizing agents, and cannot be depended on to form a passive protecting film or when there is a certain amount of abrasion to be contended with.

The two most important desiderata in galvanizing are as follows: Firstly, intimate contact of the two metals. Secondly, as large an exposed surface of zinc as possible. The first can be attained, in a measure, in hot galvanizing by mechanical pressure, such as cold-rolling or wiping, and accounts for the

relatively high efficiency of properly wiped galvanized wire, band and strip metal. In most hot work spaces are left here and there between the zinc and the article coated, and oxidation will slowly proceed, out of sight, but none the less surely. The use of sal ammoniac as a flux is responsible for much of this class of damage.

The second desideratum is attained by giving a rough rather than a smooth surface to the zinc coating. A finely granulated surface may appear smooth to the eye, but it is rough in the sense which I mean.

Hot and cold galvanizing are too well known to describe again; I shall only refer to their distinguishing features when comparing them with the new process of dry galvanizing, which it is the purpose of this short article to discuss more especially in its theoretical aspects upon which must be built time, quality and cost improvements if such be possible.

Sherardizing owes its existence to the peculiar properties of zinc dust. Zinc dust (commonly known as "blue powder") is a by-product of the smelting furnaces; in the distillation of spelter from blende (Sphalerite) the amount of dust which sublimates in the flues will amount to perhaps 5 or 10 per cent of the yield of spelter. In this dust there are beads of pure zinc and a small amount of free oxide, besides traces of cadmium, lead, iron, etc., but it is mostly composed of im-

palpable particles of a blue-grey powder from $\frac{1}{40,000}$ to $\frac{1}{50,000}$

of an inch in diameter.

Every result I have so far obtained indicates that zinc dust is zinc in a very unstable state, due to the sudden cooling to which the minute particles have been subjected; each particle is a tiny zinc "Rupert's drop." The chilled surface of these apparently perfectly spherical particles is undoubtedly oxidized, and this is a very important fact in considering the theory of zinc-dust action. Inside, the molecules of the metal are packed without regular order, it is sub-crystalline, and these molecules are striving to adjust themselves in a more comfortable manner; this, however, cannot be done without bursting the shell in which they are confined, and the reaction of this catastrophe results in vaporization. Hence, at a temperature far below that at which the metal itself would be volatilized, the zinc dust is converted into a vapor of zinc with oxide.

The temperature at which zinc volatilizes is 940°C ., and its melting point is 419°C ., yet the zinc dust breaks into vapor in the neighborhood of 200°C ., which, it is interesting to note, is about the temperature at which metallic zinc is so brittle that it can be powdered in a mortar. Zinc vapor reacts with sulphur. Twenty-five years ago Schwartz found that zinc dust and sulphur reacted when subjected to percussion; it seems to me that this indicates very clearly that the zinc dust is in an abnormal physical condition. So fine is the dew of this zinc vapor that the close-adhering deposit which it leaves on the inside of a test tube is iridescent in places, showing that its structural dimensions are of the order of light wave lengths.

If a closed receptacle is partially filled with zinc dust and heated to 200°C . or more the dust will break down into vapor or gas under considerable pressure. If metallic articles are immersed in the dust they will, after having been subjected to the action of the dust and its vapor for a period of 5 minutes or more, be found to have condensed some zinc upon their surface and in their pores. Zinc free from oxide will not give any result, and it must, therefore, be supposed that the oxide is reduced through galvanic action at the surface of the article, the free oxygen then combining with an equivalent of free zinc, the resulting oxide being in turn reduced at the surface of the article. Whatever the exact nature of the action may be the metal of the article seems to have its surface saturated with the vapor which is condensed after occlusion.

A light sherardizing shows a bright deposit, although it does not appear as such to the naked eye on account of its roughness, which is due to innumerable cavities or recesses which do

not reflect effectively; this brightness suggests reduction. A heavier sherardizing is silver-gray, due to an outer coating of slightly oxidized zinc particles closely packed with occasional clusters which stand out from the surface. In the process of sherardizing contact with the dust is necessary, excepting under conditions which will be the subject of a future article when I am further advanced in my investigations.

The name of Sherard Cowper-Coles is familiar to all electrolytic workers. The salient trait of this versatile English metallurgist seems to be a gift for framing original deductions from common-place observations; in other words, he is a "genius." The presence of some zinc dust in an annealing pot, and the silvery blemishes which an ordinary person might have passed over, if not without comment, at any rate without deduction, is responsible for a very surprising and valuable commercial application of the principles enunciated in the opening paragraphs of this article, principles which, by themselves, are of the type which inspire the most off-handed pooh-poohs of our average business friends.

In practice, sherardizing (as it is named after its inventor) is performed as follows: The articles to be coated are placed in a suitable retort—usually a cast iron drum—and covered with commercial zinc dust; the retort is closed as tightly as possible, and even luted, to prevent the egress of the vapor, which is at a higher pressure than the atmosphere of the oven (not to keep the air out as generally supposed).¹

The retort is now placed in an oven and heated to a temperature of 300°C . or more; it has been customary to go by the heat of the oven, but this has little bearing on the process, it is the heat inside the retort which counts; the time during which the retort or drum will be left in the oven will depend on the deposit which it is desired to get. The retort is allowed to cool by natural means and the articles are taken out. By having two drums in connection with each furnace the operation can be made continuous. Drums can be revolved continuously or intermittently, but when the objects are not touching one another there is no advantage whatever in changing their positions.

At first sight it seems strange that the drum itself should not be coated; this is due to its higher temperature; it emphasizes the fact that we are in the presence of a case of condensation like that of atmospheric moisture on a cold water-pipe. In the patent drums used for electrolytic work, motion is necessary to coat the object evenly; in sherardizing each particle of dust may be likened to an anode from which zinc ions emanate. Vapor and osmotic pressures are, with few exceptions, mathematically identical, my comparison of the two processes is, therefore, not without some justification, and it will help the conception of what happens in sherardizing to keep in mind the properties of electrolytes. In one case the electrical energy is supplied from without, in the other case it is supplied from within.

Mr. Cowper-Coles has suggested that sherardizing may be due to cataphoresis (electric osmosis). There may be some truth in this, but judging by my own experiments I cannot see any more than a very natural mechanical action, combined with catalytic effects which are by no means uncommon. I have found that no action takes place unless the vapor be under pressure in a closed receptacle; not only is it impossible to coat a piece of metal immersed in dust contained in a dish and placed over a burner, but the presence of an unlimited supply of oxygen will cause oxidation of the dust in preference to its disintegration; if oxidation is out of its power the dust will break into vapor, the energy required for this being evidently greater than what is needed for producing oxidation.

The higher the pressure the more effectively will the zinc gas force itself into the metal, and the temperature at which the article should be to receive its deposit most rapidly depends, in my estimation, on the relative specific heats of the zinc vapor

¹ I have successfully operated a small retort with a hole in it through which the vapor blew out and ignited; there was no possibility whatever of air getting in.

and the metal of which the article is composed, at their working temperatures. Hydrogen, which is produced by an acid pickle, becomes occluded in the iron, forming a brittle alloy, and the gaseous component of this alloy can only be removed by baking; zinc gas acts in very much the same way, and once condensed it cannot be removed by any temperature short of the point of ebullition of zinc, which is 940° C.

There is no doubt about the combination of the zinc and the metal of the article being sufficiently intimate to be classed as an alloy. Copper which has been sherardized shows a skin of hard brass, and the blow-pipe is unable to form a bead of zinc on the surface of sherardized iron as it does when the iron has been hot or electrically coated; the temperature required to fuse the zinc-iron alloy is above that produced by the ordinary blow-pipe.

Micrographs show that there is a blending of the metals, and the old surface is lost and cannot be recovered by either chemical or mechanical means. It is never required that galvanizing be removed, so that there is no disadvantage in this fact, but very much the reverse, because there can be nothing more desirable than that the protection be due to the nature of the surface itself.

The longer the process is conducted the deeper will the vapor go into the metal, but in addition to this, condensation will take place on the surface in closely series granules, and will increase the exterior coating of practically pure zinc which, with increasing purity, will grow softer as it thickens. The pressure in the retort is practically constant at constant temperature; as it is relieved by condensation of the vapor, the hottest dust particles break into vapor and re-establish the equilibrium.

Electro-galvanized work is readily buffed, but the polish is evanescent; it is too soft to stand any reasonable amount of wear, and it rapidly dulls by oxidation. This dulling is mainly due to the zinc being compelled to oxidize in order to protect the iron which is being continually attacked by the air which reaches it through the pores. All other conditions being equal, the oxidation of electro-galvanized coatings is a function of their porosity; spongy coatings, due to bad management, whiten with surprising rapidity. There are no pores in a sherardized surface, but apart from this the polish which can be given to the zinc-iron alloy is very nearly as hard as nickel-plating, and is very much superior as a protection. If nickel-plating is preferred it can be done over sherardizing instead of over copper. Polished sherardizing is of a better color than nickel-plating, it is bluish like silver and its reflecting power is higher; the difference in this respect is striking to the naked eye, but even more so when the two are viewed side by side under the microscope.

Commercially, sherardizing possesses many vital advantages over the older methods. The first cost of the plant is lower and the depreciation is very slight; it is quite simple and requires no special engineering skill. Weight for weight the zinc deposited costs less and it is more efficient. Irregular shapes and recessed pieces receive just as even a coating as the most regular. There are no residues such as dross, ashes and remnants of anodes.

The labor charges are small, and unskilled men can do the work, the most exacting requirement being the ability to read a pyrometer to within 50° C. Less care is required in cleaning the articles to be sherardized; oil will not produce blisters as in hot work, nor dust pin-holes as in cold work; in fact, oily articles receive a somewhat better finish. Threads of screws, bolts and nuts do not have to be recut and are thoroughly protected. Articles can be put into the retorts wet. The process can be worked intermittently and started quickly on account of the low temperature employed.

There is a considerable saving in fuel over the hot process in which several tons of zinc in each pot must be kept in a molten condition day and night and fifty-two Sundays a year. The cost of fuel is not as great as the cost of current in the electric process. Large articles do not have to be brought to

the temperature of the zinc dust; in fact, the temperature of the articles has very little to do with the excellence of the result. No flux is needed, but the addition of a little charcoal is found to be beneficial; no supplies are used which in the matter of cost can compare with electrical supplies.

There is no danger whatever from explosion, no broken castings nor distorted iron work to replace. Curiously enough the temper of the finest steel blades is in no way injured, and sword blades which have been sherardized have stood the British army test of being bent back to the hilt. The temperature of sherardizing corresponds to pale blue, and it might be expected that, not being unlike sand tempering, all articles would be drawn to a blue. The truth of the matter is that we know as yet very little about what is going on inside the retort, and especially so in regard to temperature. The temperature of vaporization is so low and the condensation so rapid that it appears doubtful to me if the articles ever reach anywhere near the temperatures which have entered into our discussion. It is a case where we can readily mold our theories to agree with the results of experience.

Sherardizing is drawn down with the article—be it sheet, wire or strip—to which it is applied. A valuable feature of the new process is its adaptability to a great many objects which have not hitherto received such reliable protection from rust on account of the impossibility, either physical or economic, of doing so by means of zinc. Take fly-screening: hot galvanizing would fill the meshes and electro-galvanizing consumes too much current to apply it to such fine work. Harvester cutters and innumerable machined articles could be enhanced in value 100 per cent or more by having their lives lengthened by alloying the surface with zinc (galvalloying?).

An electrolytic flashing with zinc is valueless save for the inspection of surface flaws. A flashing by sherardizing is an efficient rust preventative. Slightly sherardized boiler tubes for the export market reach their destination uninjured by the sea air. Silverware which has been flashed before being polished will not tarnish through the action of H_2S . Sherardized aluminium can be readily soldered. In either case there is no change of appearance.

A very interesting application of sherardizing has been found in the production of damascened work of a novel and artistic kind. The process is simplicity itself. The article is coated with a stopping-off material, which is then removed wherever inlaying or onlaying is desired. It is then placed in an airtight box containing the dust of the metal which is to be alloyed, and the whole is heated for 10 minutes or longer, the tints and shading off at the edges depending on the duration of the operation, which resembles case hardening, excepting that metallic dusts take the place of carbon.

That surface alloying for the purpose of protecting against oxidation will some day be applied to large pieces, such as structural material and steel cars, is the author's firm belief, and for those who have not read between the lines of this article, he will publish the experiments and reasons upon which he founds his belief in some future number of the *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

THE MODERN THEORY OF ELECTRICAL CONDUCTIVITY OF METALS.

From my March letter there was unavoidably held over for lack of space a reference to the paper which Prof. J. J. Thomson read upon this subject before a crowded audience at the Institution of Electrical Engineers. In a form of the theory which the author published some few years ago he limited the movement of the particles to the case of negative particles. He then supposed that the actual carriers of the electricity in the

metal were those small negatively electrified systems which he calls corpuscles, and which are called by others electrons (or electrons by Lord Kelvin), and that the heavier positive particles took but little or no part in the conduction of the electricity. This view has the advantage that the passage of electricity through a metal is not accompanied by any transport of the metal, a result which has often been looked for but never detected.

In the usual form of the theory, which I shall show later on requires some modification, the structure of the metal is somewhat as follows: By the action of one atom of the metal on another, corpuscles are split off from the atoms of the metal and they remain diffused through the mass of metal; so that we may picture to ourselves a metal somewhat like a porous body, the pores of which are occupied by a substance with the properties of a perfect gas. In the older theory it was supposed that these corpuscles remained free for a time sufficiently long to enable them to get into thermal equilibrium with the metal itself, so that, like all gases, the average kinetic energy of the corpuscles was a constant merely depending upon the temperature.

With all gases, the average kinetic energy is the same if the temperature is the same. As these corpuscles are exceedingly small compared with the molecules of hydrogen, the mass of a corpuscle being only about $1/34000$ th part of that of a molecule of hydrogen, the corpuscle must move with very much greater rapidity than the hydrogen molecule at the same temperature; in fact, that the square of the velocity of the corpuscle must be 3400 times the square of the velocity of the hydrogen molecule, with the result that the velocity of a corpuscle at 0° C. would be about 10^7 centimeters per second, or, roughly speaking, about 60 miles per second.

These rapidly moving corpuscles are supposed to be disseminated throughout the metal, and to be moving with this velocity in all directions. If there is no external force acting upon them, though there is this movement of electricity through the metal, yet there is no transport of electricity in one direction rather than in another. There are as many of these corpuscles moving in the one direction as in the opposite, so that there is no resultant flow of electricity.

But if we apply to these corpuscles an external electric force, then the corpuscles drift under the action of the force, drifting since they are negatively electrified, in the opposite direction to the force, so that a definite flow of these corpuscles now takes place through the metal, carrying with them the proportionate amount of electricity, and it is that flow which, according to this theory, constitutes the electric current.

Further on Prof. Thomson suggested that "the electric force, instead of acting on the corpuscles after they have left their atoms, really acts upon the atoms before the corpuscles leave them. Instead of the atoms acting on each other, think of a system of electric doublets acting on each other, and the corpuscles flows from one into the other. If you have all the doublets arranged higgledy-piggledy in the metal, then there will be as many corpuscles going one way as there are going the other; there will be no transport through the metal.

"But supposing the action of the electric force is, to polarize these doublets before so as to drag them into line, very much on the old view of Grotthus's chain and the old theory of electrolysis; supposing, for example, you drew them all into line with the negative ends pointing in one way and the positive in another, you would get a transport. When these corpuscles jump they would jump the same way, and you would get a definite transport of electricity through the metal. I think the modification that is required is to suppose that, when the electric force acts upon the metal, what it does is to arrange to a certain extent these atoms or doublets which are acting upon each other into a line, so that when they discharge the corpuscles from one to another, these corpuscles go in a definite direction."

Finally the Hall effect was discussed and explained in the following terms: "Suppose that we have a doublet with nega-

tive and positive ends, then if this is started moving by being pulled round by an electric force, these ends start moving in opposite directions; and if there is a magnetic field acting upon the thing at right angles to the electric force, then there will be forces acting upon these two ends. If those ends move with equal and opposite velocities, the forces on the two ends are equal; but if the velocities are different, then the forces on the ends are different and there is a couple produced; in fact, this thing, when placed in a magnetic field, behaves exactly like a gyroscope.

"Suppose that you had a pendulum, with a gyroscope bob instead of the ordinary bob; suppose it were in a position inclined to the vertical, corresponding to one of your doublets before an electric force acts upon it; supposing now you let it fall; gravity will not pull it straight down to the vertical as soon as the gyroscope begins to fall, but will cause it to spin round like an elliptic pendulum, one way or the other, according to the direction of rotation of the bob. So that if you had this gyroscope properly attached to a doublet; then when you attempted to pull it along the lines of electric force it would come out a little at right angles, and there would be a current either that way or the other way, according to the sign of what corresponds to the spin of the top of the gyroscope. So that, in addition to this movement along the direction of the line of force, you would get these doublets tilting up a little bit at right angles to it; there would be a polarization in the direction of right angles to it, and that polarization would produce a current in that direction; so that unless the negative and positive ends move with equal and opposite velocities—that is, unless the centre of gravity of the thing is exactly midway between the negative and the positive charge—then these molecules, when they are placed in a magnetic field, will be acting like gyroscopes, and if you attempt to pull them by a force in one direction, they tend to squirm off in a direction at right angles to it; and that, I think, accounts for the Hall effect in metals."

Only four speeches followed—probably because the specific inductive capacity of the brains of the members of the I. E. E. is very low in regard to physics. Lord Rayleigh and Dr. Glazebrook praised the paper. Prof. Silvanus Thompson asked sundry questions, and Sir William Preece seconded Dr. Glazebrook's proposal of a vote of thanks. In reply, Prof. J. J. Thomson dealt with some of the questions propounded and declared for the term corpuscle instead of electron, using the word corpuscle to denote a negative electron, and calling a positive electron by the double-worded name. The only drawback to the use of the word electron in both cases, was that it suggested an equality in their properties. For the rest, the speaker's exact words will bear quotation without any condensation:

"Then Prof. Silvanus Thompson went into some questions which if I could answer I should be very near solving the problem of the universe—the relation between electricity and matter, and whether a corpuscle was a bit of electricity or a bit of matter with a charge on it. I do not know what electricity is and I do not know what matter is. I have tried to state what I regard as energy. Taking the corpuscle, I think that all the energy possessed by a corpuscle is due to the magnetic field that it produces when in motion, or kinetic energy. I should like to ask those people who talk about electricity and matter to try to think for themselves what they mean by matter and what they mean by electricity. If they did so, they would not find it so easy to define the terms they mean. Then with regard to the question of the action of one atom upon another, I myself regard an atom as a collection of positive and negative electricity, not exactly neutral, but somewhat analogous to a magnet with positive and negative poles."

MARKET PRICES IN MARCH.

The following closing prices have to be noted in the chemical trade for the end of March:

Copper sulphate, £33.26 per ton; shellac, £9.11 per cwt;

bleaching powder (35 per cent), £4.10 per ton; ammonia sulphate, £12.26 per ton. The coal tar products are slightly weaker, crude carbolic acid (62 per cent) being quoted at 1s. 8d. per lb., and benzole (90 per cent) at 7s. 3/4d. Cyanides (98 per cent) are weaker at 7 1/2d., having been at 8 1/2d. on March 15. No changes are reported in regard to the soda products.

Copper prices have been an interesting study. Opening at £108.15 per ton, the high price of £110.10 was reached on March 12. Prices then fell away to £106 on the 19th, rallied to £107.5 on the 21st, and then, under pressure of the general fall in securities, receded to £97.10. Tin opened at £192, and remained in this neighborhood until the 14th. The price then broke to £186, and after a slight recovery receded further to £182 on March 25, rallying to £184.15 on March 30. English lead maintains its characteristically stolid level, opening at £20 per ton and closing at £20. Pig iron prices have been fairly well maintained. Opening at 55 per cwt. it has closed at 53.2 1/2 per cwt., intermediate extreme prices being 57 on the 4th and 53.3 on the 13th. Quotations on March 30 were as follows: Hematite pig iron, 77; steel rails, £6.15; steel plates, £7.10 to £7.12.6.

As to the future, the large firms are said to be fairly well occupied, and good dividends have been paid. It is generally considered that the recent "flurry" in prices of raw material is due to the panic in certain securities and has no real basis in the reduction of manufacturing demands. Nevertheless, it must be remembered that the recent half prices of tin and copper have been a first-class incitement to increased production from new enterprises, and that supply in any period of trade activity does ultimately catch up with the demand, and then, outstripping it, inaugurates a period of depression.

LONDON, April, 1907.

SYNOPSIS OF PERIODICAL LITERATURE

METALLURGY.

PRECIOUS METALS.

The Computation of the Crushing Efficiency of Tube Mills.—The discussion of the paper of Pierce and Caldecott, which was recently abstracted in the synopsis, is reported in the *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, January, 1907. Mr. E. Laschinger refers at length to a series of grading tests carried out with the product of the tube mills. For a further practical study of the problem of crushing efficiency he makes the following suggestions: 1. A grading analysis should be conducted with a sufficient number of screens. 2. All grading should be done with a set of standard screens having equal differences of aperture between them. The results of such grading would at once indicate the nature, as regards fineness, of the product which the crushing machines will produce under different conditions of operation, and what is the kind of product that may be expected from that class of machine. This information would be most useful in the designing of plants inasmuch as such crushing machinery could be installed as would give the highest percentage of that size of grain which is amenable to the treatment process decided upon. Furthermore, if the tube mill product were graded with a standard set of screens having apertures of 0.04, 0.035, 0.030, 0.025, 0.020, 0.015, 0.010 and 0.005 inch, it would be easy to tell by inspection the law of crushing for the tube mill when working under the conditions obtaining when the samples were taken. If the law of crushing were that of equal weights of particles of successive diameters, the percentage weights of each grade would be equal. It would also be at once apparent what particular grade was mostly being produced and it might be possible, between certain limits, by regulating the pebble load, rate of feed and speed of the mills to produce different de-

grees of fineness of reground product at will. 3. It is, of course, necessary to have a maximum aperture screen, slightly larger in aperture than the battery screen, so as to first eliminate from the sample of the material going to the tube mill any accidental particles due to broken screens, resetting of tiers, etc., and to eliminate from the sample of material leaving the tube mill worn out pebbles coming through the discharge plate. The slimes in the sample should, of course, be washed out before grading. 4. It is problematical whether woven screens with an aperture much less than 0.005 inch are at all reliable, because crimping and weaving seem to be irregular and the fine wires are easily displaced, thus giving great variations in apertures.

Cyanide Works Clean-Up Practice.—In the discussion of the paper by Mr. S. E. Thomas, formerly abstracted, W. R. Dowling (*Journal of the Chemical, Metallurgical and Mining Society of South Africa*, January, 1907), calls attention to the loss of gold which takes place at the clean-up or dressing of extractors owing to the high value of the solution leaving the boxes at these periods. He has overcome the possibility of the samples becoming salted by sampling the sumps periodically over the 24 hours and assaying daily at clean-up time as well as at other times through the month. When sampling an individual box over the first 3 hours of its run after clean-up, results as high as 3 dwt. per ton have been found, as high indeed as the solution entering the box. Half dwt. values over 24 hours are common. Dowling believes that most of the value carried over is from mechanically suspended gold, which appears to redissolve quickly after leaving the box and results from small, rich particles loosely adhering to the returned zinc becoming detached by the stream and chemical action. The system of circulating the solution to the box or pumping from the tail of a newly started box reduces the evil, but is not altogether satisfactory. He suggests the installation of an additional filter press, through which all solution leaving the newly started boxes should be pumped. To the solution before entering the press should be added enough zinc fume to precipitate any gold in solution as well as to keep the suspended gold from solution. The author also refers to some work he did in pot smelting in the ordinary reverberatory furnace with the object of reducing the loss of dusting in handling the calcined slimes received in the preparation for smelting. The preparatory operations consisted in first, crushing in a coffee mill; second, mixing with fluxes, and, third, transferring to the pots, and gold slime dust was produced in each of these. The loss of gold dust was avoided by the following procedure: After calcining and cooling, the tray of slimes was weighed and the previously ascertained weight of the tray deducted. The requisite amount of manganese dioxide was evenly spread over the surface of the slimes, and by means of a small, flat shovel with sides the slimes with complement of manganese dioxide were transferred to the pots. Any large lumps of slimes were crushed carefully in the pots by pressure only of a pestle. The borax and sand were added on top of the slimes and the pots were ready for fusion. On pouring, the slag was liquid, and after cooling and breaking up did not show any unfused lumps. The bullion was satisfactory and there was no excess of rills of gold in the slag, while the mixing in the pot during fusion appeared to be perfect.

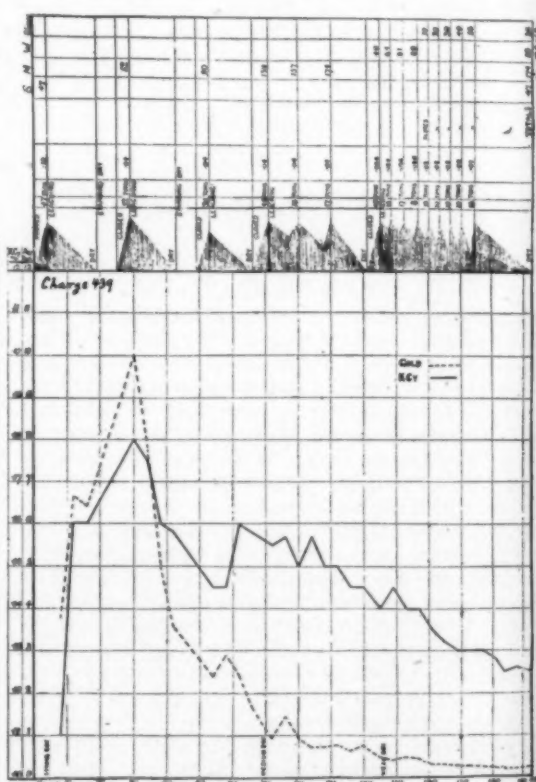
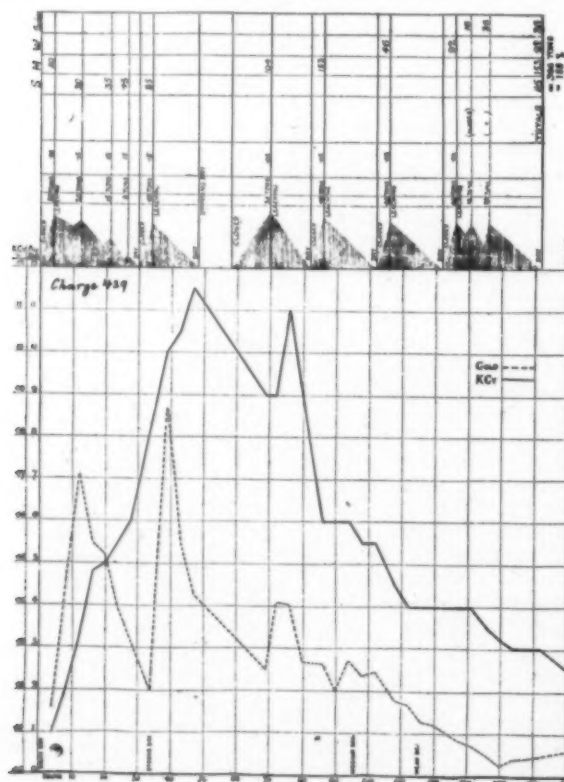
Colloidal Silicic Acid in Slimes.—W. A. Caldecott, in a communication to the *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, January, 1907, attributes the difficulty during treatment by the decantation process of settling accumulated slimes or slimes from partly weathered pyritic ore when compared with current slimes from freshly-mined pyritic blanket ore to the formation of colloidal silicic acid. He assumes that the free sulphuric acid liberated by weathering pyrites decomposes any silicates of alumina or magnesia present with the consequent liberation of silicic

acid or colloidal silicic hydrate. Thus, kaolin would be decomposed as follows:

$2\text{AlH SiO}_3 \cdot \text{H}_2\text{O} + 3\text{H}_2\text{SO}_4 = \text{Al}_2(\text{SO}_4)_3 + 2\text{Si}(\text{OH})_4 + \text{H}_2\text{O}$. The colloidal silicic acid formed in this way remains suspended for long periods in water or alkaline solutions in a gelatinous, spongy form and thus retards settlement of the non-colloidal slime particles. Similarly on leaching or filter pressing it retards the passage of solutions, but dehydration by drying, calcination or roasting corrects it into the non-colloidal state so that settlement or leaching can then go on freely. The writer adds that experiments made by adding a small percentage of artificially prepared silicic acid to current pyritic slimes showed a remarkable decrease of the rate of settlement in comparative trials. As kaolin is derived from the decomposition of feldspar, he considers it possible that clayey ore may likewise contain colloidal silicic acid liberated from the alkaline silicate present in the feldspar, by carbonic acid liberated from the air, and that the plasticity of clay is also partly due to this cause.

Extraction of Gold from Cyanide Solutions.—The extraction of the values obtained on the Transvaal ores must be considered as second to that on no other goldfield in the world, and the Rand has been justly famous for its advanced metallurgical practice. Efforts are, however, being constantly made to improve the extraction still further, it having been calculated then an improvement of even as little as $\frac{1}{2}$ per cent would represent a sum of more than \$500,000 on the usual gold out-

effected by the method of continuous leaching, and the most profitable compromise between it and the former one has to be determined experimentally by taking into account the cost and the values of the solutions leaving the tanks under the different possible methods of treatment. Mr. White gives the accompanying two diagrams, showing the gold and cyanide value of solutions, leaving two fairly representative charges of the same total weight of tonnage of solution (366 tons of solution to 230 tons of sand). The method adopted in charge 439 was based upon observation of results from charge 429, the endeavor being made (1) to reduce the cyanide consumption and (2) to keep the gold values in correspondence with the cyanide strength so that all the rich solution might be leached into the strong extraction boxes without the necessity of putting too weak solution in cyanide through these boxes or too strong a solution through the weak boxes. He points out that the diagram of charge 429 shows that the practice of commencing with a weak or medium solution is not to be commended except in special cases, not now under consideration. The first leachings on this plan begin to carry gold very soon and the cyanide strength lags behind too far. These leachings would have to go into weak boxes, and the author thinks that they are largely responsible for the formation of the objectionable white precipitate. In his view the precipitation of the latter depends upon no definite factors such as alkalinity or cyanide strength as far as actual practice is concerned, but solely upon the rate of change of these factors. A certain



put of the Rand. Mr. H. A. White, before a recent session of the Chemical, Metallurgical and Mining Society of South Africa (*Journal of the Society*, February), dealt with the possibilities of increased extraction from the cyanide leaching solutions. In chemical operations the greatest washing effect is obtained in the case of a precipitate by allowing each portion of wash water to drain completely and to use no more than sufficient to cover the precipitate. This method, however, in the case of leaching cyanide sands cannot be adopted as it requires too much time. A certain amount of displacement is

amount is held in solution definitely corresponding to alkalinity, or in the form of a double cyanide or zinc and potassium dependent on the cyanide strength. Temperature is also an important factor. Any reduction in the strength in these constituents will cause zinc compounds to be precipitated, and the aim should be to effect this in the tanks and not in the boxes. If these premises are true then it follows that the first drainings from the sand tanks, which carry practically no cyanide or alkalinity, must not go to a weak box where they mix with solutions nearly saturated with zinc, and so reduce their pro-

tective strength to the point where zinc hydrate falls out. These solutions must go to either strong or medium boxes, where their lack of strength is counterbalanced by solution flowing from other tanks under treatment and the danger point is not reached. To effect this with minimum loss the new tank must be started by a saturation with strong solution, and a gain of time is also effected by this method, as appears from a comparison of the diagrams.

Treatment of Auriferous Ore Containing Insoluble Arsenides.—In a paper read before the Chemical, Metallurgical and Mining Society of South Africa, published in the *Journal of the Society February, 1907*, Mr. J. K. Wilson outlines experiments undertaken with arsenic-bearing tailings of an auriferous South African ore, with a view to reduce the cyanide consumption in their treatment. The arsenic contents of the tailings are due to arsenical pyrites in the ore, the bulk of the gold being free and fairly coarse and amenable to amalgamation. No concentration is attempted, as the quantity is too small and the value of the concentrates too low. Both accumulated and current tailings are treated by the cyanide process. An analysis of the tailings shows the following composition: SiO_2 , 36 to 40 per cent; Fe_2O_3 , 40 to 43 per cent; Fe_3O_4 , nil to 2 per cent; Al_2O_3 , 11.8 to 12.5 per cent; arsenic as arsenide, 2.05 to 4.7 per cent; S, 0.2 to 2.0 per cent; Sb, nil to heavy traces; Cu, nil to traces; acid salts, nil to 0.087 (H_2SO_4). Experiments, which are set forth in detail, were conducted to ascertain the action of the ore on solutions of alkali as well as on cyanide solutions with the following results: (1) The consumption of lime by the ore is due principally to the presence of arsenic which exists in the form of arsenides insoluble in water. (2) These arsenides act slowly on the alkalis, decomposing them, and the stronger the alkaline solution the more rapid and complete is the reaction, which takes place. (3) Cyanide solution also dissolves these arsenides and is decomposed by them. (4) Arsenic may be practically precipitated from cyanide solution by the addition of $\text{Ca}(\text{OH})_2$ in excess, some of the cyanide being regenerated, and such precipitate is more or less soluble in excess of neutral or slightly alkaline cyanide solution. (5) The presence of zinc, arsenic or other base metals in cyanide solution retards its solvent power for gold. From these facts it seemed feasible, provided that the compounds formed with lime and arsenic were less soluble in cyanide solution than the arsenical compounds already existing in the ore, to apply the lime to the ore first and to allow a complete reaction to take place before adding the cyanide solution, and that by thus proceeding the latter would be less liable to be attacked than if the lime and the cyanide solution were added together as had been the method of treatment adopted up to the time of the experiments. The method was, therefore, changed along the lines suggested and has led to greatly improved results. In the case of accumulated tailings the requisite amount of lime is fed into each truck as the vat is charged, the charge being then saturated with water and allowed to soak for about 24 hours. The water is then drawn off and cyanide solution of a strength of from 0.04 to 0.06 per cent is run on; this strength of solution is maintained throughout the treatment, and after each wash the charge is leached dry. The required amount of solution is about $3\frac{1}{2}$ to 1 of ore; the time of treatment is from 8 to 12 days. When current tailings are treated the requisite amount of lime is added in the mill. The results of treatment under the old method were as follows: Amount treated, 1,832 tons; assay value before treatment, 3,997 dwt. per ton; assay value after treatment, 1,461 dwt. per ton; total gold contents of tailings, 364.29 oz.; total gold contents of residue, 133.83 oz.; theoretical recovery, 230.46 oz., equivalent to 63.26 per cent. Fine gold actually recovered, 194.68 oz.; estimated gold remaining in zinc, 25.00 oz.; actual recovery, 53.44 per cent; cyanide consumed per ton, 1.95 pounds; lime consumed per ton, 13.50 pounds; average time of treatment, 12 days. The improved results on adoption of the new method are

shown by the following data: Amount of tailings treated, 1,853 tons; assay value before treatment, 4,40 dwt. per ton; assay value after treatment, 1,143 dwt. per ton; total gold contents of tailings, 411.36 oz.; total gold contents of residues, 105.99 oz.; theoretical recovery, 305.37 oz., equivalent to 74.23 per cent. Fine gold actually recovered, 300.61 oz.; actual recovery, 73.03 per cent; cyanide consumed per ton, 1.45 pounds; lime consumed per ton, 11.00 pounds; average time of treatment 9 days.

Removing Broken Stem Ends from Stamp Heads.—The use of an hydraulic press for removing broken stem ends from heads instead of the old practice of knocking the ends out by hammers or blowing them out by dynamite is gradually being adopted and is no doubt much more advantageous than the former methods. Sometimes, however, especially with heavy stamps, the stems stick very tight. Mr. Q. C. McMillan, in the *Journal of the Chemical, Metallurgical and Mining Society of South Africa, February*, describes his method for removing such stems which would not move with a pressure of 140 tons. He applied the heat of a blow lamp, so as to expand the metal sufficiently to enable the press to do the work. The application of the blow lamp for a few minutes was sufficient to enable the press, whose limit was 140 tons, to squeeze out the broken ends with half the pressure previously applied unsuccessfully. The blow lamp has also proved extremely useful in loosening cams, both of the keyed and the Blanton pattern.

Clarifying Cyanide Solution for Precipitation.—In a communication to the *Journal of the Chemical, Metallurgical and Mining Society of South Africa, February*, Mr. W. S. Mann briefly describes the method he adopted for clarifying cyanide solutions for precipitation when treating dust from a dry crushing plant in Mexico with which he had considerable trouble in obtaining a clear solution. As settling tanks give poor results, and a sand filter was unsatisfactory owing to the fineness of the only sand available. In order to obtain a clear solution he placed common oakum, such as is used for calking tanks, in two individual zinc boxes, allowing the solution to pass through from the bottom as with the other compartments, which were filled with zinc. This procedure gave a perfectly clear solution. To clean, the solution was first turned into the second compartment. The solution in the first compartment was then drained into a bucket and passed through the second box, after which the oakum was easily and quickly cleaned by kneading with the hands and rinsing with clear water.

Treatment of Cyanide Precipitates.—Mr. L. Bullock, in *Mining and Scientific Press*, March 16, describes an interesting variation of the usual method of dealing with the product from the zinc precipitation boxes, in use at Copala, Simboa, Mexico. The precipitation boxes have eight compartments with bottom discharge valves. The slime is run into a two-compartment screening tank, and the precipitate, which contains from 18 to 20 per cent moisture is then brought into a dryer, which consists of an iron cylinder 30 feet long by 30 inches in diameter, having an external fire-box. A car, 24 feet long and holding six portable steel trays containing the precipitate, is run in on a track. The moisture is reduced in this dryer to 5 per cent and the precipitate then goes to a mixing box, where the flux and binder are thoroughly incorporated. The mixture is then pressed into briquets in a small briquetting machine of the F. W. Braun Co. Only the No. 300 crucible is used for melting, a unit charge consisting of 220 pounds precipitates with its flux. As the briquets melt down more are added until the crucible has its full charge. No dusting takes place. All zinc shorts are returned to the boxes, and thus the acid treatment is entirely dispensed with. The boxes are filled by first putting, say, 2 inches of long zinc on the trays of the first section of the precipitation boxes, then lightly laying the short zinc on top to a depth of 1.5 inches, and alternating in this manner until the section is half filled. The boxes are then filled entirely with carefully placed long zinc. The briquetting

partly does away with the expense of calcining, as the product is dried to only 5 per cent moisture. This product is claimed to be in every way safer and cleaner to handle, and no spitting or dusting occurs in the crucibles. The capacity of the latter is also greatly increased unless the unsatisfactory practice is followed of adding the dry calcined powder to the later, which obviously gives use to more or less dusting.

IRON AND STEEL.

Alloys with Arsenic.—K. Friedrich, professor at the Freiberg Bergschule, publishes in *Metallurgie* for March 8, an interesting study of iron-arsenic alloys (or compounds?) in the range from 8.4 to 56 per cent arsenic. By melting together iron powder reduced by hydrogen with a great excess of metallic arsenic 99.77 per cent pure, in a clay crucible at temperatures of 1,100° to 1,150°, alloys were produced up to 56 per cent arsenic. Remelting of this alloy with a great excess of arsenic, both in the crucible and in an electric furnace, failed to produce alloys richer in arsenic. The mineral hollingite corresponds to the formula FeAs_3 , containing 72.8 per cent arsenic, and on heating splits up into FeAs and As , the former containing 57.2 per cent arsenic. It is thus evident that Friedrich obtained synthetically quite nearly the compound FeAs .

The melting point diagram showed the following phenomena: Alloys from 8.4 to 30 per cent, arsenic showed a setting point, falling from 1,384° to 836°, respectively, all, however, showing a halting point at 830° to 836°. These data show these compositions to consist of crystals of iron, or iron and a little arsenic, with the very well-marked eutectic of 30 arsenic—70 iron, melting at 836°. With over 30 per cent arsenic the eutectic persists, mixed, however, with the compound FeAs (40 per cent arsenic), which melts at a maximum on the curve of 919°. Above 40 per cent arsenic the melting point falls slightly and then rises, evidently towards the melting point of the compound FeAs , with 57 per cent of arsenic, melting at 1,027°. The eutectic in this range has a melting point of 796°; but the data are somewhat confused by intermediate mixed crystals separating out.

Taken together, Prof. Friedrich has proved the existence of the compounds Fe^3As and FeAs , with a very strongly marked eutectic at 30 per cent arsenic, having the theoretical composition Fe^3As , or nearly FeAs . The study of the rich alloys shows the probable existence of FeAs_3 , produced by a chemical reaction during slow cooling at 800°. None of the alloys investigated liquated; those with over 40 per cent arsenic were not attracted by the magnet.

Piping.—In a 106-page paper, on "Piping and Segregation in Steel Ingots," Prof. Henry M. Howe attempts to collate all the phenomena and their attempted explanations, and also brings forward some new ideas as to the why and wherefore of a "pipe." The paper was read first by title at the London meeting and at the recent New York meeting of the Institute of Mining Engineers.

Concerning the primary cause of piping, the author infers that it is "the virtual expansion of the outer walls of the ingot in the early part of freezing." Consider the ingot while freezing to consist of a relatively cool outer shell, a much hotter inner shell and a liquid interior. The outer shell is kept larger than it would be if cooling alone, because of the resistance of the inner shells, that is, it is virtually expanded. As the inner layers cool, being welded fast to the expanded outer shell, they do not change their outer dimension but contract all away from the inside. It is this contraction of the just solidified inner layers, towards the outside layer and away from the interior, which causes the pipe. Prof. Howe elaborates this statement and analyses the reasons therefor into some fifty-two pages of print.

Shortening of the pipe may be accomplished by a number of devices, such as casting in wide instead of narrow moulds, using thinner walled moulds, or moulds lined with sand, so

as to reduce the great chilling of the outer layers and their consequently large virtual expansion. Detering the cooling of the top of the ingot tends to raise the segregation towards the top and to shorten up the pipe. Casting with the wide end of the ingot uppermost also shortens the pipe; in fact, Howe makes a strong appeal for the commercial introduction of casting in this manner. Liquid compression reduces the pipe by distributing the contraction of interior volume over the whole section of the ingot. Blow-holes take up some of the room otherwise represented by the pipe, and therefore diminish the volume of the pipe.

As regards segregation, it is pointed out that the segregate is lighter than the rest of the metal, and tends to rise in the pipe; that its rising is retarded by compression of the upper part of the ingot, but facilitated by compressing the middle, such as in Illingsworth's system. Top casting is also better in this respect than bottom casting, casting slowly and at as low a temperature as practicable has a like effect. It is recommended that for high quality steel it be specified that casting be done from the top, with large end up and with a sinking head.

ANALYSIS OF CURRENT ELECTRO-CHEMICAL PATENTS.

ELECTRIC FURNACES.

Electric Heating for Further Increasing the Temperature of Molten Materials.—F. Wynne, 848,422, March 26, 1907.

Application filed June 23, 1905.

Details of a method by which material that has been heated to a red heat or to a higher temperature and thereby rendered molten or semi-molten, can be further heated while

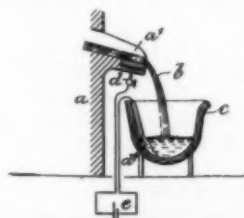


FIG. 1.—SUPPLEMENTARY ELECTRIC HEATING.

flowing from one receptacle or place to another, "as from a furnace or melting pot to a ladle, casting mold, rolling table or distributing trough or runner or from a ladle to a mold, rolling table or distributing trough or runner." The object is, of course, to reheat or superheat the molten material during its flow, so as to counteract the cooling effect of the air. Fig. 1 shows the arrangement by which a stream *b* of red-hot molten metal is electrically heated during its passage from a melting furnace *a* into a ladle *c*. The electrode *d*' in the ladle wall and a suitably-arranged electrode *d* in the spout of the melting furnace are connected with a source *e* of electric energy.

ELECTROLYTIC PROCESSES.

Battery of Electrolytic Cells.—Robert H. F. Findlay, 850,867, April 16, 1907. Application filed Jan. 12, 1907.

The cell is divided by means of two diaphragms into three compartments. The middle compartment is called the electrolyte chamber. The two other compartments contain the electrodes, and are called the anode chamber and the cathode chamber. The electrolyte chamber is supplied with electrolyte through a suitable inlet and preferably under regulated pressure. The electrolyte passes from the electrolyte chamber through the diaphragms into the anode and cathode chambers, which are fitted with suitable outlets for the products of decomposition of the electrolyte. In practice, the inventor builds up a number of such cells into a single battery, mounting them within a frame somewhat after the manner of an ordinary filter press with tightening gear to press the several cells together. When the cells are built up into a battery in this

way, instead of providing each cell with a containing vessel or case he sometimes finds it convenient to machine off the sides of the plates constituting the electrodes and the diaphragms at their outer edges and puts in distance-pieces at these parts, so that the spaces within these distance-pieces and between diaphragms and electrodes shall constitute the chambers or parts thereof hereinbefore referred to. In a preferred form he makes the anodes of a number of rods or other shaped pieces of carbon, clamped or otherwise held by plates of wood or other insulating material and the cathodes of plates of iron, preferably provided with grooves, corrugations or their equivalents. The pressure in the electrolyte chamber is made greater than that in the anode or cathode chamber.

Anode for Electroplating Cells.—George M. Elliott, 850,823, April 16, 1907. Application filed May 7, 1906 (assigned to Zucker & Levett & Loeb Co.).

The anode has at its top a transverse bar or head from which depend a series of teeth or columns which represent the anode proper. Attached to the upper side of the head are a plurality of ears or lugs by which the anode is suspended within the cell. The teeth of the anode are slightly tapering toward their lower ends.

DISCHARGES THROUGH GASES.

Nitric Acid from Air.—Albert Neuburger, 850,392, April 16, 1907. Application filed Dec. 24, 1904.

One of the great difficulties in producing nitric acid by means of electric discharges through air is due to the fact that in general nitrous acid is produced together with nitric acid. The present inventor states that the percentage of nitrous acid diminishes with the diminution of the power employed within the primary circuit (which contains the primary of a transformer, the secondary of which is connected to a spark gap). "If one were to employ, for example, a power of 200 watts and over, a mixture of equal parts of nitric and nitrous acid would be produced, so that consequently 50 per cent of the latter would be present. When using a power of 150 watts the percentage of nitrous acid sinks to 30 per cent. On using 120 watts it amounts to only 18 per cent. In this manner a gradual diminution of the percentage of nitrous acid takes place exactly with the diminution in the power employed, and when a very small power is employed the production of nitrous acid completely ceases, and pure nitric acid is formed." The first claim relates to "the process of oxidizing nitrogen, which consists in subjecting a mixture of nitrogen and oxygen to the action of an electric arc employing a maximum energy of 120 watts."

BATTERIES.

Composite Metal.—T. A. Edison, 850,881, April 16, 1907. Application filed March 30, 1905.

The "composite metal" consists of two or more metals physically admixed in such a way that "each metal exists as a practically integral sponge-like structure, the cells or pores of either metal being completely filled by the body presented by the other metal, whereby the entire bulk of the composite structure as a whole is continually solid throughout." The mixture of the two metals, however, is not so intimate or molecular as in alloys. By treating such a composite metal with an acid which dissolves only one of the two metals, this is removed and there remains a spongy integral mass of the other metal. Mr. Edison uses such a composite metal for the production of scales, flakes or foils of nickel and cobalt for admixture with the active material in the negative electrode of his storage battery. The method of manufacture is as follows: To a solution of sulphate of nickel and sulphate of cobalt he adds a potash solution in excess of that necessary to precipitate the two salts. The solutions are then boiled, resulting in the precipitation of hydroxides of nickel and cobalt, which are allowed to settle. The solution is now drawn off and the precipitated hydroxides are washed. The mixed mass

of hydroxides of cobalt and nickel is dried and screened into granules of substantially uniform size. The granules are reduced in a hydrogen atmosphere in a heated retort. The reduced metallic granules are then subjected to successive rolling operations with oil to convert the composite metal into fine scales or flakes.

Secondary Battery Plate.—L. N. J. Roselle, 850,788, April 16, 1907. Application filed Feb. 13, 1905.

The first claim relates to "the process of manufacturing plates for secondary electric batteries, which consists in casting or molding in a mold a core of active material, casting around said core, while being in hot condition a ramified grid in a second mold, at the same time forming on the entire surface of said core between the ramifications of the grid a thin film of metal, then allowing the whole to cool slowly and finally removing the thin film of metal."

Battery Plate Holder.—W. G. C. Krause, 848,559, March 26, 1907. Application filed June 24, 1905.

Constructional details of supporting frames for the negative (specially copper oxide) plates of primary batteries. Claim 4 refers to details of a battery plate holder, comprising "the combination with a cover of a U-shaped wire frame attached thereto, a U-shaped gate member hinged to said U-shaped frame, a plate intermediate said U-shaped members and means to lock said gate in its closed position against the plate."

Battery Receptacle.—H. P. R. L. Poerschke and G. A. Wedekind, 843,549, Feb. 5, 1907. Application filed July 25, 1904.

Details of construction of receptacles for cells made of cast iron, bronze, copper or steel, the walls and partitions of which are so shaped as to afford support to the reducing agent, for example, copper oxide. The first claim refers to a "battery-container electrode, having walls of cast iron bulging outwardly to cooperate with the opposite electrode, a plurality of dovetailed-holding projections formed on the inside of said bulging walls and hardened, coherent copper oxide material on said walls engaging said projections forming a substantially flush surface."

Zinc Electrode.—C. B. Schoenmehl, 848,570, March 26, 1907. Application filed Nov. 16, 1905.

Details of mechanical construction of cylindrical zinc electrodes for batteries. The third claim refers to "a battery electrode comprising a cylindrical zinc having one or more poles formed integral therewith extended beyond the main body of the zinc, suspending-rods partially inclosed in said poles and zinc in a way to be protected by the same, and hardened shoulders upon the rods to engage the under side of the cover."

Vent Valve.—F. J. Schalow, 844,755, Feb. 19, 1907. Application filed Sept. 28, 1906.

Details of construction of a vent-valve for "wet batteries," in order to allow any accumulated gas to escape without loss of electrolyte.

Primary Batteries.—Frank A. Decker, 842,389, Jan. 29, 1907. Application filed Feb. 27, 1904.

Another patent relating to the Decker battery. The object is to provide improved mechanism adapted for readily filling and emptying a nest of battery cells, to permit the ready removal and replacement of the several cells, and to prevent leakage of current or its passage from one cell to the other, either through the supply conduit or the battery fluid contained therein." The second claim reads: "A battery comprising a plurality of cells, a duct extending into the interior of each of said cells, a main conduit, a plurality of branch conduits, each communicating with said main conduit and with a plurality of said ducts, and means for removing and replacing the respective cells and their ducts independently of the remaining cells and their connections."

RECENT METALLURGICAL PATENTS

PYRITE SMELTING.

Hot Blast.—The special feature of a hot-blast smelting furnace for sulphide ores, patented by C. W. Munson (846,498, March 12, 1907), lies in the fact that the hot blast is delivered into a confined circular air space or annular chamber which circles around the entire body of ore on the furnace hearth. In this way the air is evenly distributed against the exposed surface of the ore body instead of driving the air into the ore at separate twyer openings. The twyers are protected from the molten mass by this circular air space. The construction of the furnace is shown in Fig. 1. A suitably supported hearth is represented at 20, its wall 21 being formed with a discharge hole 22, through which the molten product passes to the spout 23 for tapping it off. The metallic portion of the furnace is supported on the wall and is composed of a plurality of removable sections, each of which is hollow to form a water-jacket. Each water-jacketed section comprises an upper portion 25 and a lower portion 26 at an obtuse angle thereto, so that when six of such sections are assembled the lower portion of the chamber enclosed by the sections flares or spreads outwardly, thus giving an enlarged hearth area with inwardly-sloping upper walls. Therefore, when ore is piled on the hearth, even to a height that will reach the angle α , the natural incline of fall of the sides of the pile or heap of ore will not reach the inner surface of the sloping walls. The inclined or flaring portion is formed with a twyer-opening to receive the air blast through tubes or nozzles 28 from a suitable source of supply of heated air, such as a hot-air box 29 surrounding the brick part 30 of the furnace resting on the metallic sections. It will be seen from the diagram that there is always preserved a free air space between the surface of the fused mass and the mouth of the twyers the blast is driven into the air space just inside the twyers and is continuous and encircles the entire ore body. The result is said to be that the blast penetrates the surface of the ore body evenly, and is distributed uniformly and carries down the values into the matte or slag.

IRON AND STEEL.

Self-Hardening Steel.—James Churchward (845,757, March 5, 1907) patents the following composition: 94.6 per cent steel (having in it 0.6 per cent carbon), 3 nickel, 1.5 chromium, 0.25 manganese, 0.5 tungsten and 0.15 vanadium. "It is believed that the alloying elements react on each other to produce chemical and molecular changes of such a nature that the tungsten, chromium and manganese are permitted to harden the steel, while the vanadium removes or prevents brittleness and imparts toughness without softening the alloy."

LEAD.

Lead Oxide.—A process of making lead oxide, for use as pigments, which is stated to be quick in operation and yield a

pure product of superior quality, is patented by John W. Bailey (846,444, March 12, 1907, assigned to United Lead Co.). The process is based on the fact that finely divided metallic lead which may be produced by blowing a jet of steam or compressed air through a stream of molten lead, can be further reduced to a very fine dust or impalpable powder by means of a suitable pulverizing or attrition mill, and can then be quickly and economically reduced to the state of an oxide by being exposed to the action of a current of hot air and moisture. The pure product thus obtained is ready for use without further treatment, such as grinding, washing, etc. The inventor describes a suitable pulverizing mill adapted to reduce comminuted metal to an impalpable powder. This powder is then introduced into an apparatus consisting of several closed muffles in series through which it is passed with the aid of agitating devices from end to end. Each muffle has one passage for air supply, and in order to supply the moisture a steam jet is arranged in each air-supply passage, so as to act at the same time to induce the current of moist air through each muffle and to suitably heat the air as it is introduced.

ZINC.

Treatment of Lead and Zinc Sulphide Ores.—To convert sulphide ores of lead and zinc, together with silver, into chlorides, it has been proposed to suspend them in a bath of fused zinc chloride or of a mixture of chlorides of zinc and lead, and to subject them therein to the action of chlorine or of sulphur chloride. But this treatment involves a loss of zinc chloride by volatilization when the fused mass is heated to the vaporizing point of sulphur, and it is impossible to treat by this process ores which contain more than 30 per cent of gangue, by reason of the lack of fluidity of the resulting pasty mixture. O. Fronek (845,868, March 5, 1907) proposes to add sodium chloride to the fused bath so as to form with part or all of the zinc chloride a double salt. Then follows the treatment with chlorine or sulphur chloride. This bath has, in comparison with zinc chloride, a lower melting point but a higher volatilizing point. This not only insures that no zinc chloride shall be lost when the temperature is raised to a point sufficient to volatilize the separated sulphur, but it permits the initial fusion of the bath to be accomplished with relatively little difficulty. The bath is comparatively very fluid and mobile. This not only facilitates the distribution of the gaseous chloridizing agent, but permits the treatment of ores containing a relatively high percentage of gangue.

COPPER.

Hardening Copper.—R. T. Anderson (845,606, Feb. 26, 1907) tempers copper articles to various degrees of hardness by placing them in white sand, surrounded by sheets of zinc, also embedded in the sand, and by heating the whole. The amount of volatilized zinc which is absorbed by the copper determines the degree of hardness of the latter.

ALUMINIUM.

Soldering.—To solder aluminium by means of tin, R. A. Hall (845,948, March 5, 1907) uses a soldering compound consisting of 60 per cent stearic acid, 25 mutton tallow, 10 pure lard, 5 paraffin wax. These are melted together, allowed to cool and granulated. In order to solder aluminium the contact surfaces are cleaned by scraping, the compound is then applied and acts as a flux, the soldering being done with an ordinary soldering iron.

Thermit Welding Process.

A very handsome pamphlet, illustrated by strikingly attractive half-tones, has recently been issued by the Goldschmidt Thermit Co. on their well-known thermit welding process and "what it offers to transportation companies," in enabling quick repairs and avoiding expensive delays. The illustrations show thermit repairs of shafts, stern-posts, propeller-bearings on steamships, thermit welds of broken locomotive frames, etc.

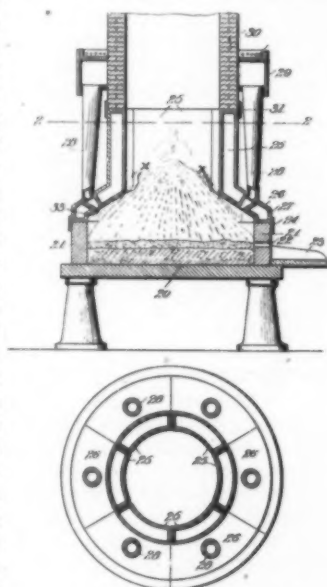


FIG. 1.—HOT-BLAST FURNACE.

Electricity at Jamestown.

Electric power for the approaching exposition at Jamestown, like that of the Buffalo Pan-American Fair, will come from a distance. Having no Niagara to rely upon, however, power for the Jamestown Exposition will be furnished by steam turbines located in the powerhouse of the Norfolk Railway and Light Co., about seven miles from the exposition grounds. This fair will be the first at which the electric power will be generated by steam turbines. The machines will be of the Curtis type, these as well as the complete electrical equipment being supplied by the General Electric Co.

The exposition authorities have entered into a contract with The Norfolk Railway and Light Co. to furnish all the electricity required for illumination and power purposes. The electricity generated at their Jamestown power-house will be transmitted on specially constructed lines to a model sub-station in Machinery Hall. Here will be located the transforming and distributing apparatus. This equipment consists of large air-cooled transformers, many smaller type H transformers for general illumination, as well as constant current transformers for the series-arc lighting system, which will be used for police illumination. At the sub-station also are motor-generator sets to provide direct current for the operation of searchlights and small motors where they may be installed by exhibitors.

The switchboard for controlling the various circuits throughout the exposition grounds is located in a gallery and is typical of modern switchboard engineering. All the electrical machinery follows standard lines similar to that installed at the St. Louis, the Pan-American and other American expositions.

Those who have seen the plans of the Jamestown Exposition predict that the electrical features, particularly the illumination, will equal, if not excel, the display at the famous Pan-American Exposition. Thousands of Edison lamps will be supplemented by search-lights both on land and on the fleets anchored in Hampton Roads, combining to make the nightly pageant magnificent and beautiful.

Works Pyrometer.

When pyrometers were first transferred from the laboratories to the works they were used in the same forms, exactly, as when in the laboratories, yet they received quite different treatment and were desired for somewhat different use. One step after another has been made to make the works instruments suitable to industrial requirements.

By far the majority of high temperature processes employ temperatures between 200° and 1,000°C and it is fortunate that thermocouples of certain metals and alloys give us stable and comparatively heavy currents within that range. By keeping down the resistance of the thermocouple, the lead and the measuring instrument, sufficient current is obtained to move the system of a double pivoted galvanometer. This means a portable or wall pattern galvanometer that can be handled as one would handle a voltmeter or ammeter, that is not affected by vibrations, and that is not expensive.

Hand in hand with the robust galvanometer comes the non-fragile portion of the pyrometer that is to go into the furnace. Either two thermocouple elements, in the form of heavy rods or one in the form of an outer tube and the other a rod inside and insulated from it make up the furnace piece. Retaining each and all of these improvements in the works pyrometer the new pyrometer, just being put out by the Wilson-Macaulen Co., 110 Liberty St., New York City, has the great added advantage of being easy to read.

A glance at the indicator (Fig. 1) shows the general appearance of it, but to realize how easy it is to read this instrument at a distance from it it must be borne in mind that the dial is 10 inches across. The scale is actually 7¾ inches long

and graduations so large that it can easily be read 12 feet away. And this is a real advantage and not only an apparent one, for we all know how much oftener things that are easy to do are done than are those which are not. When the men in the works have a great big finger continually pointing at the temperature they are forced to see and respect it. It practically forces itself upon them. When a man has to leave his work and go over and squint at an indicator he will do it just as seldom as he thinks he may. When it is so big that he can read it 12 feet away he feels that he has something to help him, not that it is an intruder that is brought in to take up his time, introduce complications and give him trouble to use.

In this pyrometer the thermocouple is in the form of a pure iron tube as one element of the couple, the other element being a rod of special alloy which passes down through the tube,



FIG. 1.—INDICATOR OF WORKS PYROMETER.

being insulated from it at all points. The tube is closed at one end and at that point the tube and rod are electrically welded.

This thermocouple makes a particularly strong fire-rod which stands an enormous amount of abuse for the reason that the insulation separating the two elements is inside the tube forming one element and so is not subjected to abrasion. In most cases the fire-rod needs no auxiliary protection, and so being exposed directly to the heat quickly assumes the surrounding temperature, causing the indicator to respond quickly to temperature changes.

The highly polished brass case with beveled plate glass front make this an exceedingly attractive instrument. The pivoting of the moving coil in jewel bearings insures lasting sensitiveness, a requisite for continued accuracy.

Notes.

Porcelain-Lined Apparatus.—Stuart & Peterson, of Burlington, N. J., have sent us their bulletins describing "Hercules" steam-jacketed kettles which they manufacture in sizes from 5 gallons capacity up to 450 gallons capacity. These kettles are also manufactured with porcelain lining in capacities ranging up to 110 gallons. If desired these kettles are provided with safety valves.

Insulator Clamps.—We have received from the Clark Electric & Manufacturing Co., of 135 Broadway, New York, several of their bulletins describing a number of types of clamps for use in connection with high-tension and cable insulators. Several types of these clamps are designed to hold firmly the transmission cable as it passes over the large petticoat insulators used in long transmission work. The bulletins also describe the copper splicing sleeves manufactured by this company for splicing hard-drawn copper wires and cables. These sleeves consist of seamless copper tubes, oval in shape, and slightly spread at each end. The ends of the cable to be spliced are slipped into the tube from its opposite ends and the splice made secure by twisting the splice from both ends.

Electrolytic Assaying.—Electrolytic processes and appa-

ratus are proving very successful and time-saving in the assay office and ore-testing works of Mr. Chas. M. Fassett, of Spokane, Wash. A motor generator, supplied with current from the lighting circuit of the city, furnishes 10-volt current for electrolytic analysis. To further reduce the voltage rheostats are provided. The modern methods by means of which it has become possible to greatly reduce the time of electrolytic analysis and which are essentially based on thorough circulation of the electrolyte, are applied. An electrolytic analysis of copper now takes 2 hours.

Calcium Cyanamide.—Before the Washington Section of the American Chemical Society, Dr. A. Frank, Jr., of Berlin, delivered a lecture, on April 1, on the manufacture and uses of calcium cyanamide (kalkstickstoff or, verbally translated, lime-nitrogen). The process is the invention of the speaker's distinguished father, Dr. A. Frank, of Charlottenburg, and Dr. N. Caro, and has been repeatedly discussed at length in our columns (see, for instance, page 77 of our March issue). Dr. Frank stated that calcium cyanamide has been produced on a commercial scale in German and Italian works for three years, and the Italian works (which were illustrated in our Vol. IV., p. 328) are now quadrupling their capacity, and in addition thereto European works are in process of construction whose initial capacity will be in the neighborhood of 50,000 tons per year. The American Cyanamid Co., of 100 Broadway, New York City, have procured the rights of the operation of the process in the United States, and are preparing for its manufacture in this country on a large scale.

The Crocker-Wheeler Co's branch managers' annual convention was recently held, and it was announced that the company had done more business during 1906 than in any other year since its foundation eighteen years ago. Among the specialties that have brought special success to the company during the past year are the W-type of rolling-mill motors, direct-current motors for machine tools, etc., and alternating-current generators of large capacity. The construction of alternating-current machinery by the Crocker-Wheeler Co. is based on the designs of Brown, Boveri & Co., of Switzerland. An instance of the recognition which they have found is the installation by the California Gas & Electric Corporation, of three 4,000-kilovolt-ampere Crocker-Wheeler alternators for operation in parallel. These are the largest alternators ever built for gas-engine drive and for operation in parallel. Some interesting data on the company's alternating-current machinery with illustrations are given in Bulletin 74, entitled *Engine Type A. C. Generators*.

Multimeter.—The Weston multimeter is a new form of electrical measuring instrument which will quite accurately serve the purposes of a direct-current voltmeter, millivoltmeter, ammeter, mil-ammeter, ohmmeter, ground detector and wheatstone bridge. It may thus be called a universal instrument of particularly high accuracy. It consists essentially of a Weston-type wheatstone bridge, a battery of twelve silver chloride cells and an indicating instrument with detachable shunts.

Héroult Electric Steel Plant at Remscheid, Germany.—The first Héroult plant at Remscheid (described on page 58 of our February issue) has been in operation nearly a year and is turning out 25 tons of high-grade steel, principally tool steel, every day. This plant was formerly a crucible-steel mill, making tool and saw steel by the old system in pots. As steel can be produced by the Héroult process much more economically and more satisfactory in quality and uniformity, the owners of the plant abandoned the old crucible pot system entirely after the Héroult process had been less than two months in operation. A new Héroult plant is now being constructed in addition to the foregoing, with a capacity of about four times the output of the present mill. Sufficient hammers and rolling mills of modern types to finish this additional product are also in course of construction. According to latest advice the new mill will be ready to start this Fall.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

CALCIUM CARBID (Continued).

No. 555,796, March 3, 1895, C. Whitehead, of Washington, D. C.

Calcines dolomite, mixes the result, double oxid of calcium and magnesium, with finely-divided carbon, and smelts with an alternating or direct current at 60 volts potential in the electric furnace, consisting of the carbon crucible electrodes set in brickwork and receiving a depending carbon rod electrode. Broken carbon in the bottom of the crucible acts as a conducting pole. The furnace has a tap hole. The described product is a double carbide of magnesium and calcium.

No. 557,057, March 24, 1896, E. N. Dickerson, of New York, N. Y.

Smelts a mixture of finely-divided calcium oxid and carbon by feeding it into an arc between the ends of two horizontal carbon electrodes. These electrodes pass closely through opposed openings in the lower end of the furnace, contact being made by rollers and springs connected with the dynamo. The charge is supported and preheated in a vertical muffle chamber, having a hopper at its upper end and containing a vertical shaft with horizontal stirring arms. Around the muffle and spaced therefrom is an outer wall through which pass blow-pipes fed with gas and air, received from heating coils, for instance, in a stack outlet at the top. The carbon monoxid rising from the zone of reduction into the space around the muffle is burned by air introduced through the blow pipes. The liquefied calcium carbide is tapped out into a car consisting of a series of molds. A flux may be added to the charge to render the carbide more fluid, depending on the distance which it has to flow.

No. 551,461, Dec. 17, 1895, William C. Clarke, of New York, N. Y.

Employs a furnace in which the electrodes are placed horizontally so that the hot gases set free will pass directly away. The charge of pulverized coke or coal twelve parts, and unslacked lime twenty parts, finely pulverized and mixed, surrounds the electrodes and nearly fills the furnace, which may have a loose perforated cover, which, however, is unnecessary. In operation the electrodes are first brought together, or nearly together, and then gradually separated, a mass of calcium carbide accumulating between them. When the ends of the electrodes nearly reach the walls of the furnace, the current is shut off, the electrodes drawn away from the carbide and the carbide, with an adherent shell of carbon and lime is removed by hooks or grappling tongs. The shell protects the carbide from the moisture of the atmosphere. The hot material remaining in the furnace falls into place between the electrodes, which are again brought together, current being turned on, fresh charge mixture added and the operation repeated. One of the electrodes may be fixed and the other movable.

No. 552,036, Dec. 24, 1895, Ludwig K. Böhm, of New York, N. Y.

A filament for incandescent electric lamps, consisting of a carbide of calcium or other metal. The filament is made from a mixture of a metallic oxid, for example, one of barium, strontium, calcium, magnesium or iron, and carbon and is then heated to incandescence by an electric current. Carbon may be added in excess of that required for the production of the carbide, the filament then consisting of a mixture of the carbide and carbon. The carbide filament, while sufficiently conductive, has the advantage of higher specific resistance than pure carbon.

(To be Continued.)

NEW BOOKS.

TRANSACTIONS OF THE AMERICAN ELECTROCHEMICAL SOCIETY. Vol. X. Tenth General Meeting, New York, Oct. 8 and 9, 1906. 132 pages. Illustrated. Bound in cloth. Price, \$3.00 net. To colleges, libraries, technical societies and journals, \$2.00 net. Philadelphia: The American Electrochemical Society.

GERMAN MONOGRAPHS ON APPLIED ELECTROCHEMISTRY. Vol. XXV. *Deutsches Patentrecht für Chemiker*. By Dr. Julius Ephraim. 608 pages. Paper cover. Price 18 marks (retail price in New York, \$6.00). Halle a. S.; Wilhelm Knapp.

A TEXTBOOK OF ELECTROCHEMISTRY. By Max Julius L. Le Blanc. Translated from the fourth enlarged German edition by Willis R. Whitney and J. W. Brown. 338 pages. Illustrated by tables and diagrams. Bound in cloth. Price, \$2.60 net. New York: The Macmillan Co.

TECHNISCHE ANWENDUNGEN DER PHYSIKALISCHEN CHEMIE. By Dr. Kurt Arndt. 304 pages, 55 illustrations. Paper binding. Price, 7 marks (retail price in New York, \$2.35). Berlin: Mayer & Müller.

THE ELEMENTS OF CHEMICAL ENGINEERING. By J. Grossman. Illustrated. Bound in cloth. Price, \$1.50 net. Philadelphia: J. B. Lippincott Co.

ELEMENTARY CHEMISTRY. By Sir Enfield H. Roscoe. Revised and enlarged by L. Elliott Brookes. 276 pages. Illustrated. Bound in cloth. Price, \$1.00. Chicago, Ill.: Frederick J. Drake & Co.

PRINCIPLES OF COPPER SMELTING. By Edward Dyer Peters. 612 pages. Bound in cloth. Price, \$5.00. New York: Hill Publishing Co.

HYDROMETALLURGY OF SILVER. With special reference to chloridizing roasting of silver ores and the extraction of silver by hyposulphite and cyanide solutions. By Ottokar Hofmann. 345 pages, 83 illustrations. Bound in cloth. Price, \$4.00. New York: Hill Publishing Co.

A TEXTBOOK OF MINING GEOLOGY. By James Park. Illustrated. Bound in cloth. Price, \$2.00 net. Philadelphia, Pa.: J. B. Lippincott Co.

THE AMERICAN STEEL WORKER. A twenty-four years' experience in the selection, annealing, working, hardening and tempering of various kinds and grades of steel. By Russell E. Markham. Second edition. 370 pages, illustrated with diagrams. Bound in cloth. Price, \$2.50. New York: Derry-Collard Co.

CEMENT AND CONCRETE. By Louis Carlton Sabin. 683 pages, 161 tables of tests. Illustrated. Cloth binding. Price, \$5.00 net. New York: McGraw Publishing Co.

THE LABORATORY BOOK OF MINERAL OIL. By J. A. Hicks. Illustrated. Bound in cloth. Price, \$1.00 net. Philadelphia: J. B. Lippincott Co.

LONG DISTANCE ELECTRIC POWER TRANSMISSION. A treatise on the hydro-electric generation of energy; its transformation, transmission and distribution. By Rollin W. Hutchinson, Jr. 345 pages. Illustrated. Bound in cloth. Price, \$3.00. New York: D. Van Nostrand Co.

PEABODY ATLAS. Shipping lines and coal railroads in the central commercial district of the United States, accompanied by chemical, geological and engineering data. By A. Bement. 149 pages. Illustrated. Bound in cloth. Price, \$5.00. Chicago: Peabody Coal Co.

QUESTIONS AND ANSWERS FROM THE GAS ENGINE. 274 pages. \$1.50. Cincinnati: Gas Engine Publishing Co.

TECHNOLOGICAL AND SCIENTIFIC DICTIONARY. By G. F. Goodchild and C. F. Tweney. 874 pages. Illustrated. Bound in half morocco. Price, \$6.00 net. Philadelphia, Pa.: J. B. Lippincott Co.

A POCKET DICTIONARY OF TECHNICAL AND SCIENTIFIC TERMS. 128 pages. Leather binding. Price, 50 cents. New York: E. P. Dutton & Co.

BOOK REVIEWS.

THE PRINCIPLES OF COPPER SMELTING. By Edward D. Peters, Professor of Metallurgy, Harvard University. Large 8vo.; pp. viii + 610. Price, \$5.00. Hill Publishing Co., New York, 1907.

The 15,000 copies of *Modern Copper Smelting*, by Dr. Peters, which have been read with instruction and pleasure by metallurgists everywhere, will have prepared a cordial reception for this supplementary volume on "reasons why," which is no less a masterpiece than its predecessor. The earlier book lays the foundation for the later one; it described apparatus, furnaces, machinery, and explained the construction and use of all the *metallurgical equipment* of a copper plant, while this deals with the chemical and physical principles involved in the operations or utilized by the apparatus, the reasons why things are done or are practicable in one way and not in another, in other words, it furnishes and co-ordinates the *mental equipment* of the copper metallurgist.

The above being stated, the nature of the book under review will be easily understood. It is didactic throughout, frequently conversational, a heart to heart talk of master to pupil. There is hardly anything to match it in modern scientific literature except Ostwald's now famous *Conversations on Chemistry*. The chapter on first principles of smelting is so clear that almost any 10-year-old boy could grasp them; the principles of roasting, chemistry of smelting, practical study of slags, description of matte, production of blister copper, refining of copper are without exception handled in the same happy style. Prof. J. W. Richards has written for this work a 25-page chapter on the thermochemistry of copper metallurgy, illustrated by several problems worked out in detail, showing how the heat is developed in copper furnaces and where it goes to.

On the practical side, there are given brief descriptions of the various furnaces and apparatus used, such as was necessary to form the basis for a discussion of the different principles utilized by the various types. Particularly important is the 125-page chapter on pyritic smelting, for the writing of which Dr. Peters possessed a *congerie* of qualifications which no one else anywhere could match. This chapter alone is a splendid contribution to the present metallurgical literature of copper.

At such a banquet, it is hypocritical to criticise minute shortcomings when there is not a single large one to point to. Still, being invited to such criticism by the author, we will point to a few differences of opinion.

We think that chemical calculations, such as are explained on page 55 and exist all through the book, are much better understood by practical man or student, if based simply on the numbers representing the atomic and molecular weights involved, and not reduced to decimal factors. We would prefer to pass from the weight of Fe to that of FeO, for instance, by multi-

plying by $\frac{72}{56} = \frac{9}{7}$, that is, by multiplying by 9 and dividing by 7, whereas, Dr. Peters prefers to multiply by 1.2857, which is

$\frac{9}{7}$ run out to four places and the fifth ignored. Curious to

relate, Dr. Peters remarks that the same result may also be attained *approximately* by using the factor $9/7$! In our view, the fraction is simpler and easier; and, of course, it is the decimal factor which does the *approximating*, if any is done at all.

In the practical study of slags, we would have preferred to see the *quantivalence* of the acid and basic elements compared, instead of the oxygen ratios. A bi-silicate is fundamentally a silicate in which the quantivalence (combining power) of the silicon present is twice that of the basic elements present, and it is these relative combining powers which constitute the essential nature of the silicate. Is it not, therefore, more logical and quite as easy arithmetically, to say that since one molecule

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of SiO_2 weighs 60, and contains one atom weight of Si whose quantivalence is 4, that, therefore, the combining power of the silicon in any weight of silica is represented by $4/60$ of the weight of the silica?

Similarly, the combining power of the

Aluminium in any weight of $\text{Al}_2\text{O}_3 = 6/102 \times \text{weight of } \text{Al}_2\text{O}_3$.

Calcium in any weight of $\text{CaO} = 2/56 \times \text{weight of CaO}$.

Magnesium in any weight of $\text{MgO} = 2/40 \times \text{weight of MgO}$.

Iron in any weight of $\text{FeO} = 2/72 \times \text{weight of FeO}$.

And thus the real things themselves, the relative combining powers of the elements present, can be compared. We plead for this way of teaching silicate ratios, because it is more fundamental, more logical and not a whit more complicated arithmetically.

The metallurgical fraternity of America has surely put an extra feather into its cap by this achievement of our genial Dr. Peters.

* * *

THE PROCEEDINGS OF THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA. Vol. III. May, 1902-June, 1903. Illustrated. 483 pages. Price, \$6.00. Johannesburg, Transvaal: The Society.

The second volume of the proceedings was reviewed on page 75 of our Vol. II., and since that time we have had very often occasion to call attention to the high value which the work of this Transvaal society must have for anybody interested in the metallurgy of gold.

The issue of the third volume had been delayed by the Boer war; the manuscripts had gone astray and had to be collected again with much trouble. We have further to acknowledge a delay of this review, which could not be avoided on account of lack of space. Even at this late date a brief summary should now be welcome. It is to be hoped that the Society will be able in future to bring out more promptly the volumes of their proceedings.

The main part of the volume (372 pages) contains the following papers with discussions:

Notes on the analysis of cyanide solutions, by Andrew F. Crosse. Notes on the treatment of slimes by means of filter presses, by Clement Dixon. The smelting and refining of zinc-gold slimes, by E. H. Johnson and W. A. Caldecott. A slight improvement in extractor boxes, by S. B. Hutt. An improved washbottle for quantitative work, by E. H. Weiskopf. Residual products of the dynamite factory and their value to the gold industry, by W. Cullen. The thermo-hyperphoric process, by A. T. Firth. Notes on valuing a gold mine, by T. Lane Carter. Notes and queries, by H. T. Durant. The lead smelting of zinc-gold slimes, by P. S. Tavener. Notes on mine sampling of the main reef series, by D. J. Williams. The economic use of petroleum oil gas furnaces, as applied to smelting, laboratory work and drill heating, by David Laird. The theory of miss-fires and some conclusions of practical value, by E. H. Weiskopf. Notes on commercial cyanide of potash, by A. Whitby. Some notes and suggestions on miners' phthisis, by W. Cullen. The regeneration of working cyanide solutions where zinc precipitation is used, by Andrew F. Crosse. Notes on the common practice of quartz milling on the Rand, by Fraser Alexander. Extraction of gold from cyanide house slimes by a wet method, by John Fleming. An automatic sampler for tailings, sands and slimes, by C. H. Pead. The refining of lead bullion (by the Parkes process), by F. L. Pidington. A few remarks on banket formation, by A. R. Sawyer.

An appendix of about 100 pages contains the minutes of the proceedings at the monthly meetings, reports, correspondence, notes, abstracts and a good index.

A complete set of the proceedings of this Society should be in the library of anybody interested in the cyanide process and in the metallurgy of gold in general.

PRODUCTION OF ALUMINIUM AND ITS INDUSTRIAL USE. By Adolphe Minet. Translated by Leonard Waldo, S. D. 8mo. 250 pages. \$2.50. John Wiley & Sons, New York.

The German edition has already been reviewed in this journal (December, 1902, p. 147). The present book is a translation, with the addition of a chapter by Minet, in which he tries to make an answer to the criticism which his work received from a German electrometallurgist, and another chapter by the translator, in which an endeavor is made to supply the glaring deficiency in the book respecting the production of aluminium in America.

The work contains six lines on the subject of the occurrence of aluminium in nature, within which compass are two mistakes, one in the composition of the most abundant ore of aluminium. Further on, the process in which Americans are the most interested, that of Chas. M. Hall, by which all the pure aluminium at present made in America is manufactured, is described in thirty-seven lines, with a reproduction of the patent drawings made in 1886—although commercial details of the present method of working have been published for ten years, even accompanied by photographs of the Hall apparatus in actual use. There are 34 pages devoted to Minet's own processes, which possess a certain historic interest but are nowhere at present being used, or at all likely to be.

The chief value of the book lies undoubtedly in the fact that it is the only book on aluminium which stands for a theory, accepted by such an authority on aluminium as Dr. Heroult. Undoubtedly, Dr. Waldo, who has also paid much attention to the electrolytic manufacture of aluminium, also believes in the same theory. For otherwise there would be no real excuse for the English edition. But it should be pointed out that the majority of the specialists on the theory of aluminium, among them Prof. Richards and Prof. Haber, are strong opponents of this theory. Whatever may be the truth, as long as there is any uncertainty on the subject, Mr. Minet had a right to define his theoretical standpoint clearly in this monograph.

We are sorry to have to state frankly that the translation is by no means satisfactory. To anybody who reads the translation and who knows the high accomplishments of the gentleman who is named as translator on the title page, it will at once be evident that he cannot have prepared the translation himself. He should have taken more care, however, in revising it, since he will be held responsible by the reader.

In many places the blunders in translation make the English quite unintelligible. Caustic soda is called several times "caustic hydrate of soda;" on page 2, "Thonerde" is translated "clay," instead of "alumina;" "rothglut" and "weissglut" are translated by the awkward expressions "red-glow" and "white-glow," instead of "red-heat" and "white-heat;" on page 7, *entwässer* (dehydrated) is translated *dephlegmated*—an alchemistic term for rectifying spirits; an amalgamated zinc plate is called an *amalgamation* of zinc plate; electrolytic coatings are called *incrustations* (page 60); using platinum electrodes and great current densities is translated platinum electrodes of great current densities; a current of 3,000 amps. is called a *stream* of 3,000 amps.; on page 21, *Wechselstrom* is translated direct current and *Gleichstrom* alternating current, just reversing the sense of the statement; *ersetzen* (to decompose) is in many places translated *dissolve*, thus completely obscuring the meaning (pages 65, 66, 68, 69, etc.); *Treibprosses* (a cupellation or scorifying operation) is translated *winnowing*, and set forth as a means of separating aluminium from zinc alloys (!) (page 69); *zerkleinert* (broken up small) is made incomprehensible by being translated *diminished* (page 54). A praiseworthy effort to correct a mistake in the original was made on pages 85 and 86, where the original erroneously spoke of the *chlorine* (instead of the *fluorine*) coming from aluminium *fluoride*, and gave the proper reaction for the fluoride; but the translator remedied the matter by speaking of the *chlorine* coming from the *chloride*, and yet retained the proper reaction in which only the *fluoride* was concerned.

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The original German book appeared in the *Sammlung Goeschen*—a collection of text books of small size and exceeding low price. Both size and price have naturally been somewhat increased in the American edition, without, however, diminishing in the least its usefulness as a very handy pocket-book for students.

The information given is clear, concise and to the point. A very large amount of material has been condensed into the small space at disposal.

Chapter I. deals with "work, current and voltage," on the basis of thermodynamics, with applications to gases and solutions. Chapter II. discusses "the chemical equilibrium, statics and kinetics," and gives the law of mass action and "van't Hoff's equation."

Then passing over to electrochemistry proper, the author deals in Chapter III. with the "theory of electrolytic dissociation," and in Chapter IV. with "conductivity," with a full explanation of the migration of ions. Chapter V. deals with "electromotive force and the galvanic current," giving the Gibbs-Helmholtz formula and van't Hoff's equation, Nernst's theory of concentration cells, etc. The sixth chapter is devoted to a discussion of "polarization and electrolysis," and gives Faraday's law. Chapter VII. is a brief exposé of the electron theory.

The summary given above hardly does justice to the enormous amount of information gathered together in this book; and what is more, this information is not only qualitative but quantitative, great care having evidently been taken to give the most reliable figures for the physical and chemical constants. The information is therefore given in the book ready for use.

Dr. Merriam has made a remarkably good translation. The book is one of the best translated books which have ever come into the reviewer's hands out of the great number of German chemical books which have been translated into English.

THE RECENT DEVELOPMENT OF PHYSICAL SCIENCE. By William Cecil Dampier Whetham, M. A., F. R. S., Fellow of Trinity College, Cambridge. 344 pages, 5 portraits, 39 illustrations. Price, bound in cloth, \$2.00 net. Philadelphia, Pa.: P. Blakiston's Son & Co.

In the development of modern physical science there have been manifest two leading tendencies, or we may say with Wagner "leitmotive." One is the tendency towards broadest generalization on broad thermodynamical basis, without employing any molecular or atomic hypotheses. The other is the bold attempt to erect a detailed theory of matter upon the electronic hypothesis.

From the work done along both these lines of endeavor, this book brings selected chapters. While the presentation of the subject is throughout excellent, the chapters on electrons are specially well written. Undoubtedly the author was inspired by the genius loci—Cambridge.

The book contains eight chapters: on the philosophical basis of physical science; the liquefaction of gases and the absolute zero of temperature; fusion and solidification; the problems of solution (both aqueous solutions and solid solutions being considered); the conduction of electricity through gases; radioactivity; atoms and æther; astro-physics. The book brings fine portraits of Newton, Kelvin, Gibbs, van't Hoff and J. J. Thomson.

The book is extremely well written and should appeal to all professional men who wish to become acquainted to a certain degree with some of the most important of these much-talked-of and little-understood new developments of physical science.

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